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METALLURGIST

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CONTENTS

THE DISTRIBUTION OF THE IRON AND STEEL INDUSTRY IN THE USSR IN THE SEVEN-YEAR PLAN

A.B. Rosentreter

Candidate of Economic Science

The magnificent program for the development of the productive potential of the country, as outlined by the Twenty-first Congress of the CPSU, envisages the solution of important tasks related to the improvement in the distribution of the iron and steel industry. A rational distribution of the iron and steel industry will assist in the complex development of the national economy of the economic regions of the USSR. It is envisaged to increase the output of ferrous metals in eastern regions in relation to the total output in the USSR. In 1965, almost half of all the output of steel and rolled product will be supplied by iron and steel factories in the Eastern USSR.

In the course of the seven-year period it is planned to put into operation the iron and steel factories of the third metallurgical base of our country. As is known, the Karaganda Metallurgical Factory ("Kazakhstanskaya Magnitka") near Temir-Tau is under construction, and the erection of the West Siberia Metallurgical Factory has been started. It is planned to begin construction of the Taishet Metallurgical Factory, the first large iron and steel factory in Western Siberia.

The Karaganda Metallurgical Factory will utilize the iron ores of the Atasu, Kacha, and Atansorsk deposits. Until the Taishet Factory is put into operation the West Siberia Factory will be supplied with ore from the Korshunovo deposit in Irkutsk province, and later on it is proposed to arrange for the supplies of ores from Anzas and Anpalyk, Ina, Beloretsk and some other deposits in the Altai-Sayan iron-ore field. After the Angara-Pit iron-ore basin is developed, the West Siberia Metallurgical Factory will be supplied with that ore too. For the Taishet Metallurgical Factory, it is planned to utilize the nearby Rudna-Gora ores in addition to the Korshunovo ores.

Coal for coking will be supplied to the factories of the third iron and steel base from the Kuznetsk and Karaganda coal basins. The introduction on an industrial scale of the new methods of coking will make it possible to use poor-coking coals from the Irkutsk, Cheremkhovo, and other coal deposits.

The construction of the Karaganda and the West Siberia Metallurgical Factory should be completed during the current seven-year period. The first units of the Taishet Metallurgical Factory will be put into operation before 1965.

To provide for the full requirements in rolled steel for Siberia and the Far East and to eliminate the extremely long and uneconomical transportation of steel, the Taishet Metallurgical Factory will produce a wide range of rolled-steel products. The West Siberia Factory will specialize in the production of sections, and the Karaganda Factory in the production of steel plate.

The economics of the new metallurgical establishments in the Eastern USSR are shown in the Table and

TABLE. The Economics of the Factories of the Third Iron and Steel Base in the USSR

Factory	Iron content in prepared ore, %	Cost, rubles		Capital expenditure per ton of pig iron, rubles			Recovery of capital cost, years		
		1 ton of prepared ore	1 ton of pig iron	on factories	on ore and fuel base	total	on factories	on ore and fuel base	total
Taishetsk	50.6	42.6	184	1216	861	2077	5.7	4.1	9.8
Karaganda	54.0	46.5	204	1282	1092	2374	5 7	4.8	10.5
West Siberia	60.3	79.2	221	1028	1243	2271	5.2	6.3	11.5
Magnitogorsk	62.12	62.7	212	1233	742	1975	4.6	2.8	7.4

*Using local ore supplies.

**Using Sokolovo-Sarbai ore supplies.

are compared with the Magnitogorsk Metallurgical Combine (according to data by Gipromez).

It is seen from the Table that the best results will be attained at the Taishet Factory where the cost of pig iron will be the lowest in the country.

In Central USSR, in 1959-1965, the second stage of the Novolipetsk Metallurgical Factory, which will be using ores from the Kursk Magnetic Anomaly and Coking Coal from the Donets Coal Basin, will be built. It is desirable to set up, at the Novolipetsk Factory, the production of steel plate and sections, in particular steel plate for the manufacture of large-diameter tubes and bent steel sections.

In the South, the construction of the Kerch Metallurgical Factory will go ahead toward the goal of completion, on the whole, during the seven-year period. The Factory will use Kerch ores and Donets coal and will produce section steel. In addition to the Kerch Factory, it is planned to complete construction of the Krivoi-Rog (first stage) and the Alchevsk Factory.

In the Northwest, the construction of the Cherepovets Metallurgical Factory will be completed. New coke batteries, large blast furnaces, open-hearth furnaces and electric steel-melting furnaces, a group of rolling mills and several sintering plants are to be built and put into operation at this Factory.

In the Urals, the productive capacity of the Magnitogorsk, the Orsk-Khalilovsk, and the Nizhne-Tagil Metallurgical Combines as well as the Chelyabinsk Metallurgical Factory and the Sinarsk and Chelyabinsk Tube Factories will be increased. The steel output of the Magnitogorsk Combine is to increase by more than one and a half times.

The Kuznetsk Metallurgical Combine in West Siberia will be substantially enlarged. One more blast furnace, new coke batteries, a converter shop, a second blooming mill, and a new oxygen plant will be put into operation. In the seven-year period, the output of pig iron and steel will increase by 42% and the output of rolled steel by 42.5%.

In the Far East, with the object of a rational utilization of local supplies of scrap and the discontinuation of the transportation of conversion pig iron, metallurgical cupola furnaces will be built at the "Amurstal'" Factory. It has also been decided to build a machine for the continuous casting of steel at this Factory.

The modernization of several other iron and steel factories is planned for 1959-1965.

To meet the demand for ferroalloys in the iron and steel industry, the existing ferroalloy factories will be extended and new ones will be built, in particular the Ermakov Factory in Kazakhstan.

The capital expenditure on the development of the iron and steel industry over the seven-year period will amount to approximately 100 billion rubles. More than half of this sum will be invested in the iron and steel industry in the eastern regions. About 70 billion rubles will be spent on increasing the capacity of existing factories, their modernization and new technical equipment. The remaining amount is earmarked for the construction of new iron and steel factories, ore mines, ore beneficiation plants and other establishments.

The Seven-Year Plan envisages the speeding-up of the construction work of new projects in the iron and steel industry. Since ample technical means and extensive experience on the rapid construction of iron and steel plants are available, it is now possible to build even a large iron and steel factory in less than five years.

The successful completion of the tasks related to the development and a better distribution of the iron and steel industry, as implied by the resolution of the Twenty-first Congress of the CPSU, will be conducive to a new rise in the Soviet economy on its way to building a communist society in the USSR.

* * *

REDUCING THE SULFUR CONTENT IN PIG IRON *

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Blast-Furnace Production

The problem of the elimination of sulfur during the blast-furnace process arose at the time when coke was first employed in blast furnaces. When pig iron was made with the use of charcoal the slag was acid, and sulfur-containing ores were desulfurized by means of a preliminary oxidizing roasting. Therefore, the problem of sulfur elimination would be now solved if it were possible to remove the sulfur from the raw materials during their preparation for the smelting process.

Unfortunately, we can only partially remove sulfur from the blast-furnace charge at present. Thus, during the process of sintering iron ores it is easy to remove up to 98% of the sulfur contained in magnetites. The production, however, of low-sulfur coke from Donets coal still remains an unsolved problem. Because of the use of sulfur-containing coke, one has to search for methods of removing the sulfur in the course of the smelting process in the blast furnace. The main requisites for a successful desulfurization in the furnace were established long ago, they are: an excess content of CaO in the slag, an adequate mobility and a high temperature of the slag. To attain these conditions it is necessary to increase the input of fluxes and coke, but the productive efficiency of the blast furnace is then reduced.

An intense search for efficient methods of pig iron desulfurization outside the blast furnace is under way in the USSR and abroad. If this problem is solved it will be possible to operate the blast furnace with acid slags and to carry out the desulfurization outside the blast furnaces.

When discussing these problems, I.I. Korobov came to the conclusion that the use of acid slags with the subsequent desulfurization of the pig iron outside the blast furnace would result in a low-quality pig iron, and therefore, under the conditions of iron production in the South, this method of iron smelting cannot be considered as suitable because, owing to the poor viscosity of primary slags, the regular operation of the blast furnace would be upset and the losses in pig iron rejected as scrap would be increased. A poor preparation of iron ores for smelting would make the blast furnace process especially difficult.

One can quite agree with Korobov's conclusions regarding the urgency of radically improving the preparation of the raw materials for smelting, of mastering the highest possible temperature of the blast as soon as possible, of eliminating irregularities in the operation of the tuyeres, of finding the optimum conditions for charging the materials into the blast furnace and of utilizing dolomitized limestone in the charge; all these suggestions have been frequently recommended in our literature. However, the conclusion on the desirability of the operation with acid slags and the subsequent desulfurization of pig iron outside the blast furnace is premature and absolutely unjustified.

First of all, the century-long experience in the operation of blast furnaces on charcoal in all countries indicates that acid slags have never complicated the operation of the blast furnace. The descent of the stock was smooth, and a slow going or hanging were rarely encountered. The properties of acid slags ensured the production of high-quality pig iron and in no way made the tapping operation difficult. It is generally known that acid slags have more stable characteristics than basic slags. An advantage of operating with acid slags is that the amount of slag in the furnace is smaller.

The feasibility and expediency of making high-acidity slags in iron smelting with coke and of the desulfurization of pig iron outside the blast furnace was shown by N.L. Gol'dshtein †. Experimental smeltings of lean ore with the use of acid slags ($CaO:SiO_2$ = approx. 0.8) carried out abroad have been described Schumacher‡. The pig iron obtained had 0.75-1.4% Si, 0.18-0.27% Mn, and 0.33-0.38% S. By the desulfurization of the pig iron at the blast furnace and after the mixer, the sulfur content in the pig iron was reduced to a normal level. No signs of an irregular operation of the furnace during one month of operation were observed in spite of the fact that the iron ores used for smelting contained 27.1% iron on the average, and the slag yield constituted 2.44 tons per ton of pig iron. The iron losses were not greater than the iron losses with basic slags. The blast temperature was low (520-550 °C). At the same time, the amount of sulfur in pig iron as well as the content of ferrous oxide in the slag depended very much on the temperature of the hearth.

Therefore, there is no reason to think that acid slags will upset the operation of the blast furnace or will result in the production of a low quality pig iron and in an

*In the "Metallurgist", No. 1, 1959, the article, "Reducing the sulfur content in pig-iron in the course of the blast-furnace process" by I.I. Korobov, was published; this is a comment on that article.

† Stal', No. 6 (1941).

‡ Stahl and Eisen, No. 12 (1939).

increase in losses of iron as scrap; this is even more true during operation with high temperature blast.

The economic advantages of desulfurizing the pig iron outside the blast furnace as compared with the cost of desulfurizing in the furnace are also obvious. We calculated from actual data that the cost of desulfurizing the pig iron with sodium carbonate outside the blast furnace is considerably lower than the cost of the additional coke and limestone required to reduce the sulfur content of pig iron in the blast furnace to the required limits (Table).

TABLE 1

Cost of Lowering the Content of Sulfur to Required Standard Limits

Sulfur content in pig iron	Method of desulfurization	
	outside the blast furnace	during the smelting process
0.081	0.99	2.9
0.092	1.08	11.5
0.105	1.18	22.5
0.118	1.31	38.8
0.126	1.40	49.5

It is seen from the Table that there is no justification for rejecting the operation of the blast furnace with the use of acid slags and the subsequent desulfurization of pig iron outside the blast furnace. At present, however, the change to the new technique cannot be carried out because no reliable method of pig iron desulfurization has been developed. Even the most extensively tested method of desulfurization by means of sodium carbonate cannot at the present stage be accepted as suitable for a systematic and large-scale desulfurization of pig iron although it makes it possible to reduce the sulfur content in pig-iron to the required limits. All the known methods of desulfurizing pig iron in the ladle complicate the operation of pig iron tapping, make the work of furnace attendants difficult, do not permit a proper observation of the process of filling the ladle and, last but not least, cause considerable difficulties in the removal of the spent slag from the ladle.

Therefore, it is necessary first of all to develop a convenient and reliable method of desulfurizing pig iron outside the blast furnace. Unfortunately, this problem has for a long time not received due attention, the research and development work has been carried out haphazardly and no significant results have been obtained. We pointed out at the All-Union Conference of Blast Furnace Operators that in order to solve this important problem it is essential to build a central plant for the desulfurization of pig iron at one of the iron and steel works in the South, so that it would be possible to carry out tests, complete the development and work out the industrial utilization of the most suitable methods. Only then would it be possible to begin the operation of blast furnaces with the use of acid slags.

The investigations aimed at the improvement of the desulfurization processes in the blast-furnace hearth are as important as ever. In this respect, I.L. Korobov's suggestion about introducing oxygen into the blast-furnace hearth at the level of the slag notch with the object of regulating the temperature of the slag should receive careful attention. We propose that an allowable quantity of lime and magnesia should be introduced with oxygen through the slag notch into the hearth. They should be introduced immediately after the tapping of the pig iron; then the newly formed lower slag would be more reactive and would more efficiently absorb the sulfur from the pig iron which accumulated in the hearth in the period between the tappings, and thus the quality of the pig iron would improve. Probably in this connection it will be possible to reduce part of the limestone from the blast-furnace charge, and this will result in a relative lowering of the basicity of the slags above the tuyeres, it will have a positive effect on the blast furnace operation and will assist in a more efficient removal of sulfur with the gases.

Attempts were made to blow in the lime through the air tuyeres with the object of improving the desulfurizing power of the slag but this method did not, and could not, give positive results since the lime blown in through the tuyeres was taken up by the gas stream into the melting zone of the slag thus complicating the smelting process. The advantage of introducing the lime through the slag notches lies in the fact that the lime is then absorbed by the lower slag and remains in the furnace hearth.

To improve the reactivity of the lower slag it is desirable to try the charging of the blast furnace with rounds of various basicity. It is essential by carefully studying and controlling the descent of the stock, to achieve conditions under which the rounds from which the lower slag is formed have a higher basicity and the remaining rounds a lower basicity.

It is essential to improve the temperature control of the upper slag so that it will be possible to estimate the thermal condition of the hearth.

We must use all possible means for a successful combatment of sulfur in blast furnace production. We must demand that coke and coal suppliers find a means of reducing the sulfur content in coal and coke, we must improve the methods of desulfurization in the blast furnace and, last but not least, we must speed up the work on mastering the desulfurization of pig iron outside the blast furnace.

The problem of operating the blast furnace with the use of acid slags and the desulfurization of pig iron outside the blast furnace has to be regarded as a most urgent problem in the iron and steel industry.

* * *

PRODUCTION OF PIG IRON WITH THE USE OF COKE IRON

Pak Syn Nok

The People's Democratic Republic of Korea

The investigations on the production of coke iron and its application for iron smelting were started during the Korean War.

At first, by making use of the experience accumulated during the investigation of the possibility of the production of pig iron with the use of anthracite, and by utilizing the method of the manufacture of ore briquettes, we carried out some elementary experiments and thermodynamic calculations which showed that if the blast furnace is filled with briquettes made of a mixture of iron ore concentrate and coal then the iron oxide is rapidly reduced to a metallic state, provided that the strength of the briquette is retained. Therefore, one can obtain pig iron from these briquettes in low shaft furnaces (during the war we were not in a position to build blast furnaces of a normal height but there was an urgent demand for pig iron).

The workers and technical personnel of the Kim Chak Iron and Steel Factory undertook to continue the investigation of the production of pig iron in low shaft furnaces. In September, 1951, we started to make briquettes from a mixture of Musan iron ore concentrate and various coals. These briquettes were resistant to cold and high temperature and stood up well to the pressure in low shaft furnaces. They were called "reducing iron-ore briquettes."

Under war conditions, no experimental apparatus was available for testing briquettes at a high temperature and, therefore, we carried out industrial tests directly in a small blast furnace during a trial smelting operation. To strengthen the briquette we subjected it to thermal treatment under pressure. The strongest briquettes were obtained from a mixture of iron ore concentrate and coking coal. This is explained by the fact that, in the course of the thermal treatment, the coal was converted into coke, and oxides of iron were reduced to metallic iron.

We called such briquettes "iron-containing coke," and later on we gave it the name of coke iron.

As a result of the laboratory preparation of coke iron the following facts were ascertained:

1. During the coking most of the Musan iron ore concentrate is reduced to metallic iron, the extent of the reduction being dependent on the conditions of coking. At the same time the grains of the reduced metallic iron adhere strongly to the neighboring grains of iron or carbon. Therefore, the mechanical strength of the coke iron produced depends on the content of iron ore concentrate in the charge prepared for coking. Up to a certain limit of the concentrate content the mechanical strength increases and then gradually decreases (Fig.).

2. When coke iron is used there is no need for high blast furnaces since the iron contained in the coke iron is already partly reduced.

3. Coke-iron production makes it possible to utilize pulverulent iron ores.

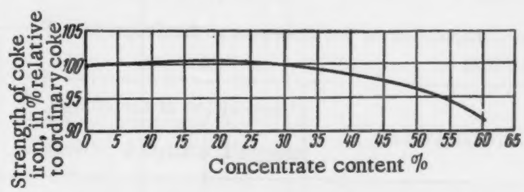

Figure. The dependence of the mechanical strength of coke iron on the content of the iron ore concentrate in the charge.

4. In the process of coking, the ore is deoxidized by methane, hydrogen and other reducing gases which are produced during the coking. The heat necessary for the deoxidation is supplied by the combustion of the coke-oven or blast-furnace gas used for heating the batteries. Therefore, less coke is used in the production of pig iron.

5. A reduction in the consumption of coke per ton of pig iron results in a reduction of blast consumption and an increase in the efficiency of the blast furnace. On the other hand, since the height of the furnace can be reduced, it is possible to improve the ratio of the working volume to the daily output of the furnace still more.

6. Under ideal conditions, when coke iron is used in the blast furnace for the production of pig iron, no other iron ores need be added, but then the content of the iron ore concentrate in the coking charge should be very high. The coke iron contains excess carbon and, therefore, during the smelting of pig iron from coke iron it is necessary to charge some additional ore and flux into the blast furnace.

After the All-Korea Conference of Scientific Workers (April, 1952) the problem of the production of pig iron from coke iron, which had been investigated for more than ten months, was discussed very extensively. Specialists in this field expressed appreciation of the work done so far.

Experimental semicommercial blast furnaces for the production of coke iron and for the smelting of pig iron were built in a region relatively safe from enemy bombardment. Large-scale tests confirmed the results of laboratory experiments and made it possible to obtain data on the process of smelting pig iron from coke iron.

The production of coke iron with an addition of iron ore concentrate to the coking charge was started in 1956, (in a coke-oven battery of 57 chambers); 14,585 tons of coke iron were produced in that period. The results of this operation showed that if the necessary operating conditions are maintained it is possible to obtain good-quality coke iron charge containing up to 60% concentrate. The silica-refractory walls of the coking chambers are not affected by the slag. The coke iron can be pushed out with the ordinary coke pusher. The composition of the charge and the properties of the coke iron produced are given in Table 1.

TABLE 1

The Composition of the Charge and the Properties of Coke Iron

Components	Composition of the charge, %		
	charge 1	charge 2	charge 3
Sankhav coal	40	35	30
Keren coal	50	40	35
Musan concentrate	10	20	30
Anthracite	—	5	5
Properties of coke iron			
Ash	15.8	16.4	17.0
Sulfur (total)	0.835	0.886	0.95
Trommel test number, %	90.5	89.5	89.63

The coking period is somewhat extended as the content of the concentrate in the charge increases. At the same time, the duration of the period depends on the conditions of coking. Therefore, it can be controlled to a certain extent by varying the conditions of coking.

During the wet quenching of coke iron, the iron becomes partly oxidized at exposed surfaces. The mechanical strength of the product is reduced. It is of great interest, therefore, to consider a dry quenching of coke iron. Our experiments in this direction were unsuccessful.

TABLE 2

Results of the Coking Operation

Items	Concentrate content in the charge %				
	0	10	20	30	40
Gas yield, m^3/ton	443	473	500	530	577
Heating value of the gas, kcal/m^3	4258	4104	4025	3819	3327
Metallic iron content, %	0	7.56	15.1	22.3	30.0
Tar yield per ton of coal, ton	0.04	0.037	0.036	0.035	0.033
Light oil yield per ton of coal, ton	0.0056	0.0057	0.0058	0.0067	0.0077
Coking period hr	18.0	18.3	18.8	19.1	19.5

The gases involved and the chemical products were studied using laboratory, semicommercial and commercial plants. We carried out a number of experiments in which the concentrate content in the charge for coking was varied from 0 to 60%. The experimental results showed (Table 2) that, other conditions being equal, the time of the evolution of coke-oven gas gradually increases as the concentrate content decreases. The yield of gas per ton of coal increases and its heating value decreases as the concentrate content in the charge is increased. The over-all heating value of all the gaseous products per ton of coal charged increases.

The tar produced during the coking is decomposed under the catalytic action of iron oxide and metallic iron, the yield of light fractions being increased. As the concentrate content decreases, the yield of tar decreases, and that of benzene and phenol increases.

The extent of the reduction of iron during the production of coke iron increases as the temperature increases and as the length of the coking period increases.

For constant temperature and constant length of coking period, as the concentrate content increases, the amount of reduced iron for the same amount of carbon decreases but its total content increases.

From January till the end of April, 1955, we made pig iron from the coke iron in a blast furnace of 73 m^3 working volume. The results of the experimental smeltings showed that the consumption of coking coal per ton of pig iron was markedly reduced. In the production of pig iron with the use of coke iron obtained from the mixture consisting of 50% iron-ore concentrate and 50% coal, the coke consumption per ton of pig-iron decreased by more than 30% as compared with the production of pig iron with the use of ordinary coke. The average daily output of the blast furnace increased by 45-50%. In addition, the gas permeability of the blast furnace charge increased significantly and the distribution of the gas stream improved. As the content of iron ore concentrate in the coke iron charge increased, the conditions for the reduction of silicon became more favorable.

Industrial tests on coke iron were made in 1958 at the Khvankhé Factory. At the beginning of July, 1958, we began the production of coke iron, and on July 23, we introduced the coke iron (made from 10% concentrate and 90% coal, instead of ordinary coke) into the charge of one of the blast furnaces at the Khvankhé Works. The concentrate had the following chemical composition, (%):

Fe	SiO_2	Al_2O_3	CaO	MgO	P_2O_5	
56.88	20.04	1.19	0.59	0.30	0.15	0.055

The coal had the following composition, percent;

				S_{tot}
10.69	26.52	62.12	0.69	0.64

Lumpy iron ore of the following composition (in %) was also used for the pig-iron production:

Fe	SiO_2	Al_2O_3	CaO	MgO	P_2O_5	S
46.93	15.94	0.54	2.21	1.70	0.136	0.032

After fifteen days operation with the use of 10% coke iron, the furnace was charged with 20% and subsequently with 30% coke iron. At present, the furnace is being successfully operated with the use of 30% coke iron. The results of the smelting operation are given in Table 3.

As a result of the operation of the blast furnace it has been established that the production of pig iron from coke iron is possible. When coke iron was used, the temperature of the inwall of the furnace and the temperature of the peripheral gases decreased from 500 to 450-470 °C. The descent of the charge was speeded up. To reduce the contents of silicon and sulfur in the pig iron it is necessary to make slags of high basicity.

TABLE 3. Operation of the Blast Furnace with the use of Coke Iron

	Coke iron		
	10% (18 days)	20% (15 days)	30% (20 days)
Average daily output, tons	488	587	640
Consumption per ton of pig iron, tons:			
coke iron	1.1	1.2	1.3
lumpy ore	1.46	1.28	1.12
limestone	0.124	0.146	0.22
coking coal	1.366	1.250	1.187
Blast:			
consumption, m^3/min	1350	1300	1330
temperature, °C	600	650	700
pressure, atmos	0.88	0.85	0.87
Blast-furnace gas:			
temperature, °C	260	250	230
CO_2 content, %	8.2	7.0	6.7
Content in pig iron, %:			
silicon	1.20	1.27	1.29
sulfur	0.043	0.048	0.052
Slag basicity $\frac{CaO}{SiO_2}$	1.05	1.12	1.20

At present, we are working on the production and use of coke iron with a high content of iron-ore concentrate. The use of this coke iron will make it possible to reduce coal consumption to 1000 kg per 1 ton of pig iron and to achieve a ratio of blast furnace working volume to daily output equal to 0.7 or less. Special attention is being paid to the possibility of using local coal and anthracite for the production of coke iron.

* * *

AUTOMATIC CONTINUOUS FEEDERS

A.P. Mal'kov

Technical Developments in the Seven-Year Plan

In 1957, the Moscow Experimental Factory for Testing Machines and Scales made experimental specimens of the LDA automatic continuous weigh feeders designed by the Special Design Office for Testing Machines of the Moscow-Town Sovnarkhoz (National Economic Council). In 1959, the Ivanovskoe Machine Works started mass production of these machines.

The automatic feeders are designated for by-product coke works and for sintering plants and are used for the continuous delivery of loose materials from bins at a predetermined rate.

The main characteristics of the feeders are given in Table 1.

The weigh feeders, however, can also be used for conveying other friable materials in addition to those listed in the Table.

The maximum throughput of a feeder can be selected by the customer from the following figures, ton/hr: 10, 12.5, 16, 20, 25, 32, 40, 50, 63, 80, 100, 125, 160.

If the feeder is intended for handling materials which are not specified in the Table, two conditions have to be complied with.

1. The material handled must flow freely onto the vibrating pan, which is 500 mm wide for feeders with a 500 or 800 mm wide belt, and 850 mm wide for feeders with a 1000 mm wide belt. When the pan is inclined at 20° to the horizontal and the amplitude of vibrations is 1.5 mm, it should be possible to exceed the nominal maximum capacity for the feeder for a long period of time, and when the pan stops vibrating the flow of materials should cease immediately. If the behavior of the materials on the vibrating pan feeder has not been investigated, one should test experimently whether the vibrating feeder can be used for the purpose. As an example we can quote the operation of the vibrating feeders when handling coal of 20 mm or less particle size, and when handling iron ore of 2 mm and less particle size. If the latter contains more than 13% moisture it hardly moves down the vibrating pan even when the feeder vibrates at the full amplitude, whereas the change in the moisture content in coal from 6-8% to 12% reduces the throughput of the feeder from 100 tons/hr to 50-60 tons/hr.

2. Each weigh feeder with a belt 500, 800, or 1000 mm wide can have two maximum volumetric throughputs determined by the depth of the material on the belt and by the two speeds of the belt (Table 2).

On multiplying the value of the maximum volumetric throughput by the weight per cubic meter of the materials to be handled, one can select the required type of feeder according to the width and the maximum capacity.

TABLE 1

Characteristics of the LDA Automatic Weigh Feeders

Mark	Width of the belt	Material handled	Weight per cubic meter, ton/m³	Maximum throughput, ton/hr.	Speed of the belt m/sec.	Dimensions, mm: without vibrating pan: length, l	width	height	with vibrating pan: length, L	width	height H	Width of the vibrating pan, mm	weight, kg: weigh feeder	vibrating pan	Total power consumption, kw.
LDA-12N	500	Scale	2.2	3	0.17										
		Blast-furnace dust	1.5	12											
LDA-32M		Limestone, dolomite	1.5	32		2600	1135	1125	4500	1135	1250	500	560	640	1.5
LDA-25N	800	Fine coke	0.5	25	0.35	2600	1435	1125	4500	1435	1350	500	610	640	1.5
LDA-60N		Tailings	1.5	60											
		Fine ore	2.2												
LDA-100N	1000	Coal	0.8—0.85	100		2600	1635	1125	4500	1635	1425	850	600	1300	2.0
LDA-13ON		Sintering charge	1.1—1.4	130											

The error in the operation of the feeder is ± 2% of the maximum value of the scale within the range of the preset throughputs, from 1/4 to full capacity.

The weigh feeder consists of a vibrating pan and a short weigh conveyor (Fig. 1).

The vibrating pan is suspended on four shock absorbers under the bin. The inclination angle of the pan is selected according to the maximum throughput required. The pan (designed by the "Mekhanobr" Institute) receives the vibrating motion from a reciprocating electric motor supplied by alternating current at 380 v (50 cps) and direct current at 20 v, the amplitude of oscillations (from 0 to 1.5 mm) being directly proportional to the dc input (from 0 to 4 amp).

Under the delivery end of the feeder pan is a weigh conveyor, mounted on a steel base; all the parts of the conveyor are assembled on a frame provided with covers which can easily be removed. The conveyor and the drive are suspended on a lever system which is counterbalanced by a spring and weights of the dustproof counterbalancing mechanism.

For a constant speed of the belt, the weight of the load on the conveyor is proportional to the throughput of the conveyor and is indicated on the scale graduated in capacity units. At the same time, the spring actuates the piston of a standard induction transmitter (stroke of the piston=5 mm). The movement of the piston corresponds to the change in the throughput of the given feeder from zero to the maximum value.

An electronic differential transformer device of the ÉPID-05 type serves as a secondary instrument which indicates and records instantaneous throughput, determines the total weight of the material delivered by the feeder, and is provided with a 3-position regulator.

This system of control is quite satisfactory and reliable in operation for freely flowing materials since deviations in the throughput of the vibrating pan when operating at a constant temperature constitute 1-2 %. Only occasionally fluctuations in the throughput are caused by the change in the weight per cubic meter of the material or in the conditions of the flow of the material from the bin.

The feeder is driven by a UL-071 universal electric motor connected through reduction gears to the shaft of the rotating transformer which supplies the dc coil of the vibrating motor through a selenium rectifier. A change of the preset throughput results in a corresponding change in the dc driving the vibrating pan and hence a corresponding change in the throughput of the vibrating pan is effected.

The electric circuit of the weigh feeder provides for the possibility of operating the driving motor at two speeds: at a low speed when the throughput deviates by 6-8% from the preset rate; and a high speed with a larger

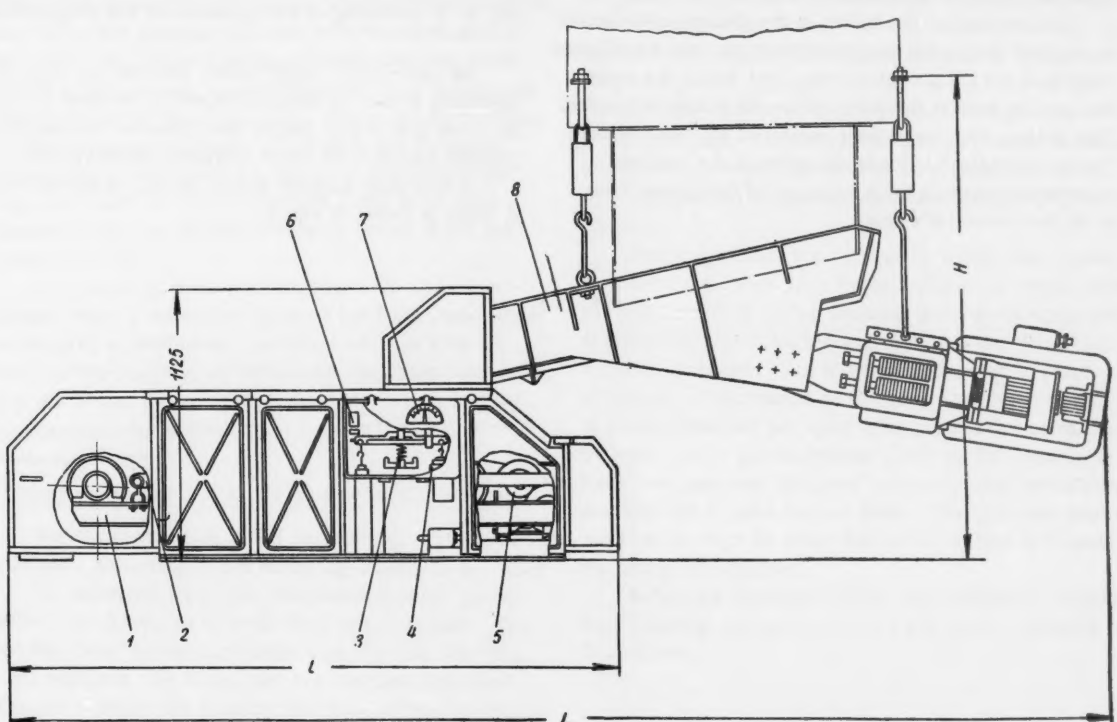

Fig. 1. Automatic weigh feeder with vibrating pan: 1) Weigh conveyor; 2) lever system; 3) spring; 4) poise weight; 5) conveyor drive; 6) induction transmitter; 7) throughput scale; 8) vibrating pan.

deviation (the high speed is mainly employed during the starting up or when the preset throughput is changed).

TABLE 2

Maximum Volumetric Throughput of the Weigh Feeder, m^3/hr

Belt width, mm	Throughput for various belt speeds, m/sec.	
	0.17	0.35
500	23	46
800	28.5	57
1 000	67	134

The weigh feeder is equipped with a signal device which closes corresponding circuits when the full range of the automatic regulation is used up, and when there is a persistent deviation of more than 2% of the actual throughput from the preset throughput.

Depending on the control system of all the mechanisms of a section where the feeders are installed, the automatic instrumentation of each mechanism can be accommodated in a separate control desk, or the secondary instruments can be accommodated in the main control room, and all starting and controlling instruments can be placed on open panels at the distribution room.

The starting of the feeder or the change in the preset throughput during the automatic operation can be effected only from the main control room, and during the repairs this can be done at the place where the feeder is located. The change from the remote control of the feeder to the "repair operation" is made by means of the control selector mounted on the front panel of the control box or on the controller's desk.

The electric circuit of the weigh feeder provides for the possibility of coupling it with other units and communication lines.

The deviations from the normal operation of the feeder can be indicated by light or sound signals.

If convenient, the ÉPID-05 apparatus can be replaced by any control system which includes an induction transmitter with a piston of 5 mm stroke and which can control the levers of the UL-071 mechanism.

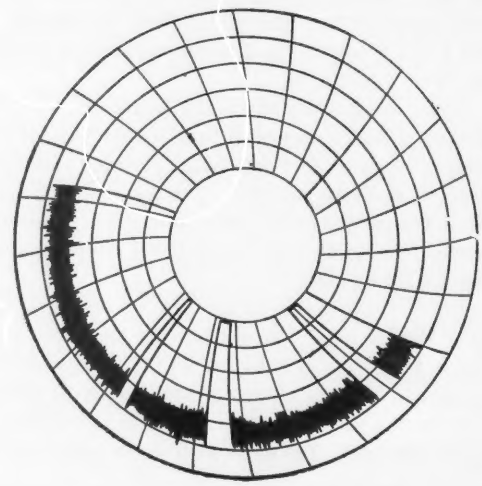

Fig. 2. A recording of the operation of the weigh feeder.

An LDA-100T weigh feeder for coal has been in operation at the Yasinovo Coke and By-Product Factory for more than a year and in that time has handled over 150,000 tons of coke while operating automatically.

A recording diagram of this feeder, taken on February 8, 1959, is shown in Fig. 2.

* * *

THE OPERATION OF A 1.5 TON BASIC INDUCTION FURNACE

A.F. Kablukovskii, I.P. Solodikhin, Ya.S. Leizerov, and S.D. Skorokhod

Making various types of steels with a low content of carbon, silicon and other elements in three-phase electric-arc furnaces with a basic lining is rather difficult. The absorption of carbon by the metal during the steelmaking process from the carbon-containing lining and from the graphitized electrodes makes it impossible to produce steel of carbon content below 0.05%. The contamination of steel with silicon in electric-arc furnaces is due to the reduction processes of silicon and the residual impurities in the ferroalloy additions. Therefore, the silicon content in chromium–aluminum, chromium–nickel–titanium, and other steels is always more than 0.25%.

Making steel in an induction furnace with a basic lining is free from all the above disadvantages and is most suitable for producing steel of low carbon and silicon contents. Until recently, induction furnaces of more than 500 kg capacity were lined with acid materials. The refractory used was quartzite containing not less than 95% silica and a very small amount of other oxides.

The range of steels produced in induction furnaces with an acid lining is limited. Some steels and alloys with a high content of manganese, aluminum and titanium, and with a low content of silicon must be made only in basic furnaces. When high-manganese steels are made, the acid lining is very rapidly damaged since manganous oxide reacts with the silica in the lining and forms easily fusible manganese silicate. Aluminum and titanium liberate the silicon from the lining, so that steel with high aluminum or titanium content made in an acid furnace is found to contain silicon in excess of the permissible limits.

As a result of prolonged investigations and experimental tests*, a refractory material has been found which has ensured a satisfactory durability (not less than 40 heats) of the crucible of the 1.5 ton induction furnace. The use of this furnace has made it possible to develop and introduce the technology of making high-alloyed steels and alloys.

Preparation and Use of a Basic Crucible

For the preparation of the basic lining of the crucible a mixture consisting of the following materials is used:

1. Industrial light-gray magnesite powder, grade MPMZ (all fractions) or MPÉP (4-2 mm fractions). The powder should contain not more than 3% CaO, 3% SiO_2, 1.5% Fe_2O_3 and 1% Al_2O_3, and not less than 85% MgO. The losses during the roasting should be within the limits of 0.3-0.5%.

2. Magnesite-chromite powder obtained by grinding (in crusher-roller mill) heat-resistant magnesite-chromite bricks of grade MKhS. For this purpose new bricks and sound parts of the bricks from old roofs of electric-arc furnaces after the removal of burnt ends saturated with oxides are used. The magnesite-chromite powder should contain not less than 60% MgO and 12-18% Cr_2O_3.

3. Caustic magnesite from the Satka deposit, containing not less than 75% MgO and, not more than 6.0% CaO, 1.5% SiO_2, 1.5% Fe_2O_3, and 1.5% Al_2O_3. The losses during the roasting should not exceed 10%, and the moisture content should not exceed 1.5%. According to standard specifications, the setting should not start until 20 minutes have elapsed and finish not later than 6 hr.

4. Ground (in crusher-roller mill) fluorspar containing not less than 92% calcium fluoride, not more than 4% silicon oxide and not more than 0.02% sulfur.

5. Ground fire clay, from the Latnaya deposit, containing 20.5-31.8% alumina and titanium dioxide, and 0.7-1.0% iron oxide. The losses during the roasting should remain within 6.9-10.4%, and the moisture content should be 5.3-8.0%.

Before it is used, the magnesite powder and ground magnesite-chromite is screened into fractions 4-2 mm, 2-1 mm and below 1 mm, and the fractions are kept in separate containers. The caustic magnesite, ground fire clay and grade I fluorspar are used in the form of fine powder after they have been screened through a sieve with 1 mm openings. The magnesite and magnesite-chromite powders are screened by hand or on mechanical sieves. The composition of the heat-resistant mixture is given in Table 1.

The components are batched by weight and mixed in special steel pans prior to the addition of water. After the addition of 5% water the mass is shoveled again until it is uniformly humidified and then it is transferred into a metal container. The mixture is prepared in batches of 100 kg. The mixture produced is considered acceptable if a lump does not fall apart after it has been compressed by hand. After eleven batches (1100 kg for 1 crucible) have been prepared the mass is covered with wet sacking and kept for at least sixteen hours. The prepared mixture must not be kept for more than 48 hr before it is used for lining the crucible.

Before the crucible is lined, the inductor is cleaned, blown through and pressed with water under a pressure of 8-10 atmos.

*A.M. Mikel'son, N.A. Shiryaev, T.B. Tatarskaya, V.G. Osipov et al. took part in this work.

After a thorough inspection and check of the tilting mechanism of the furnace and the mounting of the inductor, the internal surface of the inductor is coated with an insulating mixture consisting of quartz (66%) and alabaster powder (34%). The mass is mixed, humidified with water until it has a consistency of a thick cream and then it is rapidly applied in a uniform layer not more than 5 mm thick on the inductor. To increase its strength the layer is coated with a solution of fine magnesite-chromite powder in water glass and dried for 30-40 min with a gentle flame from an open fire laid on the bottom blocks.

Before the crucible is lined, plates of micanite (2 mm) and asbestos (4-5 mm) are laid on the bottom blocks and on the insulation layer of the inductor, and are fixed by means of detachable steel rings. The bottom of the crucible is made up of a few layers. For each layer, the mass is laid to a thickness of 25-30 mm and is rammed with a flat pneumatic rammer. Before the next layer is laid the surface of the first layer is loosened to a depth of 5 mm with a sharp rod because an insufficiently loose layer may result in the formation of cracks during the operation of the crucible.

After the ramming, a welded steel form with a bottom made of 5 m thick steel plate (Fig. 1) is placed on the furnace bottom. The plates are lap welded and the sharp edges are dressed by means of a drilling machine. For better drying of the lining, about 100-150 holes 3-5 mm in diameter are drilled in the walls and the bottom of the steel form. The form must be accurately aligned with respect to the center line of the inductor and a weight of 400-500 kg is placed inside the form in order to keep it in a stable position during the ramming of the walls. After the form has been positioned, the refractory mass is poured into the space between the form and the inductor. The walls are rammed in the same way as the bottom. During the ramming of the lining of the crucible, special care must be taken to obtain a uniform distribution of grains in the layers and to prevent the contamination with foreign matter. An accumulation of large size grains in the last portions of the lining is not permissible.

The lined crucible (Fig. 2) is kept for at least 24 hr and then is dried for 2 hr by a wood fire and for 12-14 hr

TABLE 1

Composition of the Refractory Mixture, %

Component	Particle size, mm		
	4—2	2—1	below 1
Magnesite powder	10.0	17.5	15.0
Magnesite-chromite powder	7.5	17.5	23.25
Caustic magnesite	—	—	7.5
Fluorspar	—	—	1.0
Fire clay	—	—	0.75

by a coke fire. For a uniform burning of coke over the whole height of the crucible, air is supplied intermittently from above. An intense burning of coke is not admissible since the steel form can then burn through; after the lining has been dried, the coke is removed (without tilting the furnace) and the first charge is loaded into the crucible. It is recommended that steels R9, Kh12 or carbon steel should be made in the first sintering heat.

The operating conditions for the first sintering heat in the 1.5 ton-crucible are as follows:

Power, kw:	30	50	75	100	125	150	200	250	300
Duration, min:	15	45	45	45	30	30	30	60	120

Then the full power of 550 kw is turned on; the total duration of the heat with the current on is 7 hr to 7 hr, 30 min. The diameter of the top, middle and the lower part of the crucible and the thickness of the bottom are measured after the first heat.

In order to prevent incrustation of the crucible and to increase its capacity, a few batches of steel with a high silicon content are melted. In the course of operation, the lining of the basic crucible acquires a laminar structure (Figs. 3,4). As a rule the crucible surface which is in contact with the steel becomes covered with a slag skin 5-8 mm thick. Under this skin, a layer of sintered lining 25-35 mm deep extends over the whole wall of the crucible (Fig. 3). As it approaches the inductor the sintered layer passes into semisintered (Fig. 4 b) and a nonsintered (Fig. 4 a) or buffer layer. The presence of the buffer layer and the fact that the refractory lining is not uniformly sintered throughout its depth provide a reliable protection for the inductor from the penetration of liquid metal and ensure a smooth operation of the furnace throughout the whole campaign.

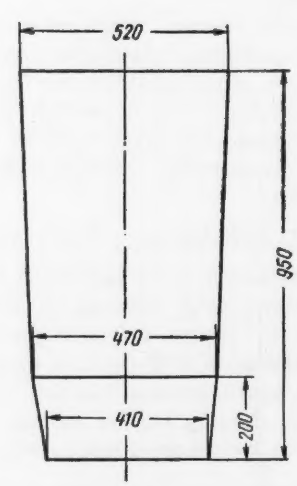

Fig. 1. Form used for lining 1.5 ton basic crucible.

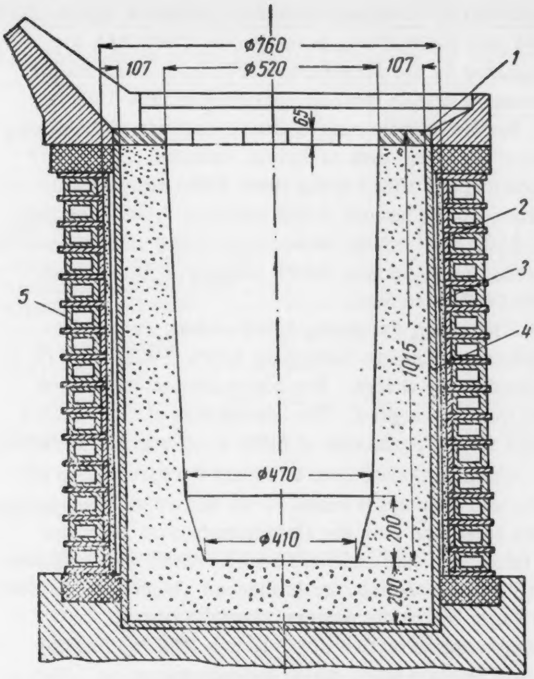

Fig. 2. Lining of a basic crucible of a high-frequency induction furnace. 1) Magnesite-chromite bricks; 2) insulation, 5 mm thick; 3) micanite, 3 mm thick; 4) asbestos, 5 mm thick; 5) inductor, 33 turns, 35×21×4 mm.

Fig. 3. Sintered layer of the wall lining.

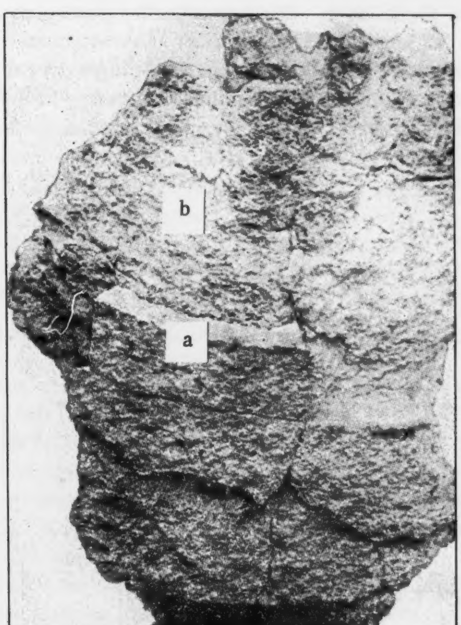

Fig. 4. Nonsintered (a) and semisintered (b) layers of the lining.

The chemical composition of the separate layers of the basic lining is given in Table 2. The low content of SiO_2, Fe_2O_3, MgO, and Cr_2O_3 and the high content of Al_2O_3 and CaO in the sintered layer as compared with other parts of the lining is explained by diffusional processes and the interaction of slag and metal with the working surface of the crucible.

Melting of Steel in a Basic Crucible

The induction furnace described here is supplied with a high frequency current from a 550 kw motor-generator set. The furnace has two melting crucibles of 1.5 ton capacity each (with acid lining), the crucibles being used alternatively. The melting in an induction furnace with a basic lining is carried out in the same way as in acid crucibles.

The first portion of the slag mixture is introduced when liquid metal appears at the bottom of the crucible. If the charge contains no easily oxidized elements (chromium, titanium, aluminum, etc.) and the finished steel contains the least amount of carbon possible, the slag mixture is added only after all the charge is melted. A mixture of 60-65% lime, 15-20% magnesite and 20-25% fluorspar is used as slag-forming material.

The components of the slag mixture must conform to special requirements. The lime must be freshly slaked and must not contain more than 0.1% sulfur. Before the mixture is prepared the lime is ground and screened through a screen with 2-3 mm openings. The metallurgical magnesite powder and fluorspar, free from im-

purities and ferrous sulfide, are screened through 1 and 2 mm openings. Before they are mixed, all the components are kept in a drying oven at 150-300 °C.

TABLE 2

Chemical Composition of the Separate Layers of the Lining, %

Layer	SiO_2	Fe_2O_3	Al_2O_3	CaO	MgO	Cr_2O_3	MnO	TiO_2
Sintered	2.20	3.20	16.10	10.10	61.80	5.11	0.02	Traces
Semisintered	1.56	3.96	2.90	7.28	72.25	7.77	0.02	»
Nonsintered	4.12	3.94	3.42	3.72	78.0	8.35	tr.	»

The slag mixture is prepared immediately before it is to be used. After all the charge in the furnace is melted the primary slag is skimmed off and fresh slag is made.

For the deoxidation of low-carbon steel through the slag during the refining period, one uses a mixture of aluminum powder and lime, which has been screened beforehand through a screen with 1 mm openings and roasted at 250-300 °C. Apart from this mixture, for the diffusional deoxidation one also uses fine fractions of coke, calcium silicon, aluminum etc. For the final deoxidation of the steel, one uses aluminum, calcium silicon, metallic calcium, nickel-magnesium alloy and other materials; the deoxiders are introduced into the steel by means of rods pushed down to the bottom of the crucible.

Several alloy steels and other alloys can be made in the basic crucible of the induction furnace. As an example we quote the process data for some resistance alloys and welding steels (Table 3).

Production of chromium-aluminum resistance alloys. ÉI595 and ÉI626 alloys, developed by TsNIIChM, are designated for the manufacture of heating elements for electric resistance furnaces operating at 1250-1300 °C.

For the improvement of the service life of the alloys, special additions, such as barium, calcium, cerium and zirconium, are added to the steel. ÉI595 alloy should contain 0.025% barium, 0.10% calcium, 0.0065% cerium and 0.060% zirconium; theoretically ÉI626 alloy should contain 1.0% barium, 0.10% calcium, 0.10% cerium, and 0.060% zirconium.

Armco iron containing 0.04% carbon, and grade 00000 ferrochromium containing 0.04% C and 74.5% Cr are used in the charge. The components of the charge are carefully weighed. The composition of the metallic charge for the production of ÉI626 alloy is given in Table 4.

Aluminum-zirconium alloy and the first portion of aluminum in lumps is added 20-30 min before the pouring of the steel, and then the aluminum-barium alloy and the remaining aluminum are added. Aluminum-calcium alloy, and ferrocerium are introduced, respectively 5 min and 2 min before the pouring. Samples of steel for forging a square section of 10-15 mm side and bending through 180° are taken during the refining period. The temperature of the liquid steel, measured by means of tungsten-molybdenum immersion thermocouples, should be between 1630-1660 °C before the pouring. The temperature drop during the pouring constitutes 50-60 °C.

The steel is poured from above into round molds to produce 500 kg ingots. Carbon tetrachloride is used to protect the steel from oxidation and to improve the surface of the ingots. One hour after the pouring, the ingots are loaded into heated boxes and transported to the forge shop. The temperature of the ingots at the time of charging into the continuous reheating furnaces should be not less than 600 °C. Some operating statistics

TABLE 3

The Chemical Composition of Some Resistance Alloys and Welding Steels

<table>
<tr><th rowspan="2">Steel or alloy grade</th><th colspan="9">Composition, %</th></tr>
<tr><th>C</th><th>Si</th><th>Mn</th><th>S</th><th>P</th><th>Cr</th><th>Ni</th><th>Al</th><th>Ba</th></tr>
<tr><td rowspan="2">ÉI 595</td><td colspan="5">Maximum</td><td rowspan="2">21.5—23.5</td><td rowspan="2">Maximum 0.60</td><td rowspan="2">4.50—5.10</td><td rowspan="2">—</td></tr>
<tr><td>0.05</td><td>0.60</td><td>0.30</td><td>0.015</td><td>0.015</td></tr>
<tr><td rowspan="2">ÉI 626</td><td colspan="5">Maximum</td><td rowspan="2">26.0—28.0</td><td rowspan="2">Maximum 0.60</td><td rowspan="2">5.0—5.80</td><td rowspan="2">—</td></tr>
<tr><td>0.05</td><td>0.60</td><td>0.30</td><td>0.015</td><td>0.015</td></tr>
<tr><td rowspan="2">ÉI 606A</td><td rowspan="2">Maximum 0.07</td><td rowspan="2">1.30—1.80</td><td colspan="3">Maximum</td><td rowspan="2">18.0—20.0</td><td rowspan="2">8.0—10.0</td><td rowspan="2">—</td><td rowspan="2">2.20—2.70</td></tr>
<tr><td>0.70</td><td>0.030</td><td>0.030</td></tr>
<tr><td rowspan="2">ÉI 903</td><td rowspan="2">0.30—0.40</td><td rowspan="2">Maximum 0.50</td><td rowspan="2">16.0—18.0</td><td colspan="2">Maximum</td><td rowspan="2">0.60</td><td rowspan="2">0.50</td><td rowspan="2">4.75—5.50</td><td rowspan="2">—</td></tr>
<tr><td>0.020</td><td>0.030</td></tr>
</table>

TABLE 4

The Composition of the Metallic Charge for the Production of ÉI626 Alloy

Material	Weight, kg	Additions, kg									
		C	Si	S	P	Cr	Al	Zr	Ca	Ba	Ce
Armco iron	575.0	0.24	1.1	0.10	0.11	—	—	—	—	—	—
Ferrochromium 00000	385.0	0.14	1.9	0.02	0.01	282	1.5	—	—	—	—
Aluminum-zirconium alloy	29.0	0.01	1.1	0.01	0.03	—	6.0	6.4	—	—	—
Aluminum-barium alloy	26.5	—	—	—	—	—	15.0	—	—	10.6	—
Aluminum-calcium	7.9	—	—	—	—	—	6.7	—	1.06	—	—
Ferrocerium	1.22	—	—	—	—	—	—	—	—	—	1.06
Aluminum (lump)	30.8	—	—	—	—	—	30.8	—	—	—	—
Total charge	1055.42	0.39	4.1	0.13	0.15	282	60.0	6.4	1.06	10.6	1.06
Element content, %	—	0.037	0.39	0.012	0.014	26.8	5.65	0.60	0.10	1.00	0.10

TABLE 5

Operating Statistics of the Production of Resistance Alloys and Welding Steels

Alloy or steel grade	Number of heats	Av. duration of the heat, hr-min.				Powder consumption, kw-hr/ton	Total loss of metal, %
		total	charging period	melt-down	refining		
ÉI 595	22	3—02	0—15	2—12	0—35	1198	5.4
ÉI 626	56	3—06	0—15	2—18	0—33	1164	5.6
ÉI 606A	61	3—08	0—11	2—26	0—31	1162	3.9
ÉI 903	10	2—28	0—08	1—47	0—33	920	4.3

of the production of the ÉI595 and ÉI626 alloys are given in Table 5.

The production of welding steels. The extensive use of electric welding in various industrial fields has created the need for the production of new types of high-alloy steel for welding electrodes.

When the induction furnaces with basic lining are used, the problem of the production of these steels becomes much simpler. As an example, we give the technology of the production of ÉI606A and ÉI903 welding steels (Table 3).

The charge used for melting ÉI606A steel consists of armco iron (565 kg), ferrochromium, grade 00000 (215 kg), and nickel or alloying scrap of Kh20N80 type (112 kg). The metallic charge for the production of the ÉI903 steel is composed of the scrap of steel 30-45 (850 kg) and metallic manganese (180 kg) containing 0.10%C and 97.0% Mn.

To reduce the loss in burning, the metallic manganese is added into the liquid metal toward the end of the melt-down period. After all the charge has been melted down the slag is skimmed off and a new slag is made; the slag and the steel are deoxidized with a mixture of aluminum powder and lime. The temperature of the steel is measured two or three times by means of immersion thermocouples during the refining period, and samples for forging into 10-15 mm side square section and bending through 180° are taken at a temperature of 1100-1150 °C.

Ferrovanadium is added to the deoxidized bath 20-25 min before the pouring. 75% ferrosilicon is used for alloying ÉI606A steel with silicon. During the melting of ÉI903 steel, the metallic titanium is added 20-25 min before the pouring and the metallic calcium is introduced into the steel 2-3 min before the tapping and before the addition of the primary aluminum. The temperature of the steel during the refining period should be maintained between 1620-1650 °C, and in the ladle—between 1570 and 1600 °C.

The steel is poured from above into round molds to produce 500 kg ingots. One hour before the pouring, the molds are coated with a solution of tar in solvent naphtha with polymerized and unsaturated hydrocarbons (1:1). The steel can also be poured into uncoated molds.

The ingots are stripped from the molds two hours after the end of pouring. To remove surface defects (scabs, folds, and splashes) the ingots are dressed on dressing machines before transfer to the forge shop. Some operating statistics of the production of ÉI606A and ÉI903 steels are given in Table 5.

The evaluation of the quality of the steel made in a 1.5 ton induction furnace with basic lining, as estimated on the basis of the accepted product, confirmed the expediency of the developed method of steel production.

The tests which were carried out showed that it is possible to replace the acid lining of crucibles of large capacity induction furnaces with basic lining.

* * *

REDUCING THE TOP-END CROP OF KILLED STEEL INGOT

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The top-end crop from killed-steel ingots constitutes 12-15% of ingot weight while the volume of the pipe constitutes only 3-4% of ingot volume. A comparison of these figures indicates the inefficiency in present methods of reducing the depth of the pipe, and of other measures aimed at a reduction of the end crop.

At the Stalino Metallurgical Factory an investigation has been carried out with the object of finding rational methods of reducing the top-end crop* by means of selecting better insulating materials, improving the design of the hot top and selecting a suitable composition of the exothermic mixtures for heating the steel in the hot top from the sides.

The selection of material for covering the steel in the hot top. With the object of investigating the effect of the method of insulating the metal surface in a hot top on the depth of the pipe, two types of insulating materials were tried out at the Factory in 1958 (Table 1). During the pouring of steel, three experimental ingots on the second or the third group of ingots from one heat were insulated by various methods (1 ingot for each method); one experimental ingot in each group was covered with the usual lunkerite used at the Factory (lunkerite 1).

When the experimental ingots were rolled at the blooming mill, transverse samples were taken from the top end at 11, 12, 13, and 14% of the length of the rolled section. The macrostructure was evaluated on the basis of the five-mark scale in use at the plant; the structure was considered unsatisfactory if the axial porosity was equal to or exceeded 2.5 marks.

The results of the investigation of the macrostructure of the hot top of ingots covered with heat-producing and insulating mixtures showed that the mixture consisting of 65% ferrosilicon, 20% potassium nitrate, and 15% fire brick as recommended by the "Serp i molot" Factory is not suitable for large ingots. When this mixture was used

TABLE 1

Composition of the Mixtures Used for Covering the Hot Tops of Ingots

Mixture	Composition							
	aluminum	ferrosilicon	fine coke	fire brick	bauxite	potassium nitrate	blast-furnace dust	blast-furnace slag
Heat-producing mixtures								
Lunkerite I	12	10	32	30	16	—	—	—
Lunkerite II	28	5	25	30	12	—	—	—
Thermite mixture*	—	65	—	15	—	20	—	—
Experimental lunkerite I	20	—	30	30	—	20	—	—
Experimental lunkerite II.	15	—	15	—	—	—	70	—
Insulating mixture								
No. 1	—	—	33	34	—	—	—	3
No. 2	—	—	40	40	—	—	—	20
No. 3	—	—	50	50	—	—	—	—

*Recommended by the "Serp i Molot" Factory

* A.A. Kazakov, I.I. Bekker, I.A. Yakunin and V.S. Terekhora took part in the investigation.

TABLE 2

Macrostructure of Ingots With Hot Tops of Volume Equal to 11% and 15% of Ingot Volume

Covering	Number of ingots	Hot top of 11% volume				Number of ingots	Hot top of 15% volume			
		11% crop		12% crop			13% crop		14% crop	
		total number of samples	no. of samples of 2.5 or higher mark	total number of samples	no. of samples of 2.5 or higher mark		total number of samples	no. of samples of 2.5 or higher mole	total number of samples	no. of samples of 2.5 or higher mole
Insulating mixture No. 3	10	10	1	8	1	20	18	4	13	3
Lunkerite I	10	6	1	10	3	20	19	7	14	2

all the samples had an unsatisfactory macrostructure, i.e., a 14% end crop did not ensure a complete removal of the pipe. Such a deep pipe in ingots insulated with thermite mixture is explained by an extremely high rate of combustion reactions and the absence of the remainder of the insulating mixture on the surface of the metal after the mixture has reacted.

Lunkerite I, which burnt slowly and left a porous layer of low thermal conductivity on the surface of the hot top, showed the best results. Judging by the preliminary results the use of this lunkerite will make it possible to reduce the volume of the metal in the hot top by approximately 1%.

Approximately the same macrostructure of the hot tops covered by lunkerite I and lunkerite II indicates that aluminum has little effect in exothermic mixtures which do not include sufficient amounts of oxygen-containing substances. Where the ingots were covered with the fire brick-coke mixture No. 3, the macrostructure of the ingot was the same as for the ordinary lunkerite. In view of the low cost of this mixture it may be recommended for use instead of expensive lunkerite.

Design improvement of the hot top. In 1958, a hot top with a smaller cross section as compared with the ordinary hot top was designed and tested at the Factory. The volume of the hot top was reduced by 1-1.5% (of the ingot volume).

Satisfactory results in the tests of this hop top led to investigations of a hot top of an even smaller cross section. The volume of the hot top was then reduced by 3.5-4% and constituted 11% of the ingot volume. The results of the study of the macrostructure of samples taken from ingots with hot tops of various volumes are shown in Table 2.

The hot tops of small cross section were practically as effective as the ordinary hot tops but with the former the metal consumption was reduced by 2%. On the basis of the experimental results a new hot top of a volume 2% smaller than the ordinary hot top was designed (Fig. 1).

The development of a method for the exothermic heating of the sides of the hot top end of an ingot. The higher the temperature of the lining of the hot top before

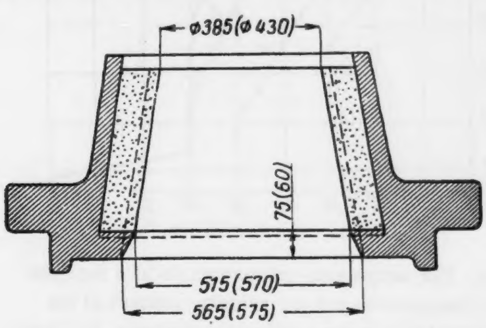


Fig. 1. Sketch of the new-type hot top (the dimensions of the old hot top are shown in brackets).

TABLE 3

Composition of Exothermic Mixtures for Hot Top Lining

Mixture No.	Composition, % (weight)						
	aluminum	scale	fire brick mortar	fine coke	potassium nitrate	chrome-magnesite mortar	heat-resistant clay
1	12	38	50	—	—	—	—
2	15	45	40	—	—	—	—
3	17	52	31	—	—	—	—
4	15	45	30	10	—	—	—
5	14	42	29	12	3	—	—
6	17	52	—	9	—	20	2
7	16	48	—	—	—	36	—

casting, the more effective the hot top is. The use of exothermic materials in the lining is equivalent to heating the lining. An exothermic lining should meet the following requirements:

1. The burning should begin at the temperature to which the surface of the lining of the hot top is heated at the end of the filling of the mold.
2. The temperature of the lining from the moment when the lining gets into contact with the liquid metal until the crystallization of the ingot is completed should be higher than the temperature of the metal in the hot top.
3. As a result of the completion of the exothermic reactions a heat resistant layer with good insulating properties should be formed from the mixture.
4. The heat-resistant layer must be easily removable together with the hot top and must be easy to remove from the hot top.
5. The use of an exothermic lining must be justifiable from the economic point of view.
6. The exothermic reactions should not impair the working conditions in the casting bay.

On the basis of the results of the tests carried out in the laboratory of the Ukrainian Institute of Metals on the use of exothermic mixtures for lining and coating the hot tops of 250 kg ingots, an investigation on the use of a mixture containing ordinary iron-aluminum thermite was undertaken. The thermite was prepared from PAP-1 or APV grade aluminum powder and iron scale screened through a sieve with 2 mm openings, the ratio of aluminum powder to iron scale being 3 : 1. The other materials were screened through a screen with 1 mm diameter openings. The composition of the mixtures is shown in Table 3. The mixtures were treated with water glass and water (1 : 1) in an amount of 6-7% on top of 100%.

The exothermic mass was applied to the lining of the hot top by means of ramming. A wooden core was placed inside the lined hot top, and the exothermic mass was rammed into the 30-40 mm annular gap by hand and partially by a pneumatic rammer. After the hot top was dried by means of a gas flame, a solid exothermic layer

which adhered strongly to the main lining was obtained. The hot top end of the ingot constituted approximately 11% of the total volume of the ingot.

During the routine heats, an experimental ingot was cast for each heat on the second or the third bottom plate into a mold in which the hot top was lined with different exothermic mixtures. Lunkerite I was used for covering the surface of the metal.

The macrostructure of the metal was tested by the same method as in the experiments with the reduced-volume hot top. The results of the experiments are given in Table 4.

The results show that the investigated exothermic mixtures, which contained iron-aluminum thermite as the combustible material, do not satisfy the conditions specified. The gain on the reduction of the end crop is less than the cost of materials and labor. Also, the fact that the ingot becomes fused with the lining indicates the unsuitability of the mixtures investigated.

TABLE 4. Results of the Tests on Exothermic Mixtures for Lining Hot Tops

Mixture No.	Number of ingots	Thermite content in the mixture, % (weight)	Height of the top top	Amount of defective metal in the hot top, percent	Condition of the lining after the casting
1	1	50	The same as for the ordinary ingot	11	The original dimensions were preserved
2	1	60	140 mm lower than for the ordinary ingot	8	Fused
3	1	70	Ditto	9	Ditto
4	1	60	The same as for an ordinary ingot	8	Fused to depth 1/3 of thickness, but original shape preserved
5	2	56	Ditto	7-8	Ditto
6	2	70	70 mm lower than an ordinary hot top	7-8	Original dimensions were preserved
7	2	64	50-70 mm lower than ordinary hot top	7-8	Ditto

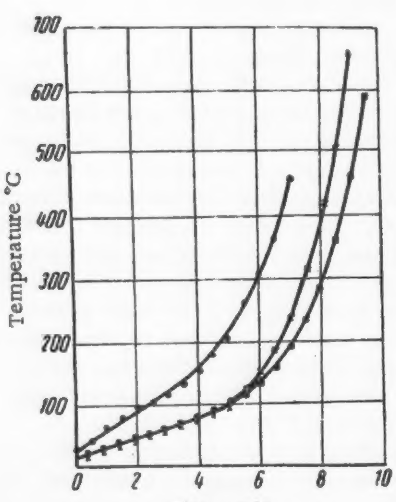


Fig. 2. The rise in the temperature of the surface of the lining of the hot top during the filling of the mold with steel.

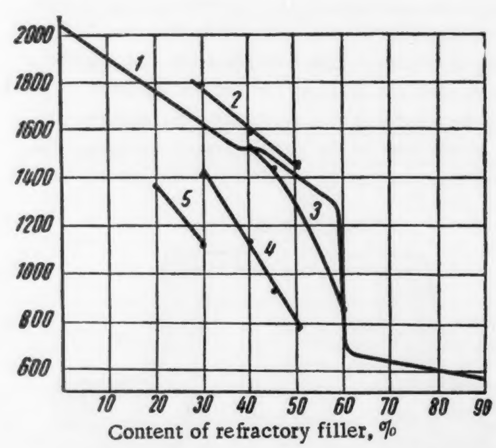


Fig. 3. The temperature of the sample as a function of the composition and the percent content of the refractory filler in the exothermic mixture: 1) Theoretical curve for the filler of a heat capacity of 0.2 kcal/kg °C; 2) magnesite; 3) builders' sand; 4) mortar; 5) ground coke.

One of the causes of the unsatisfactory results was the high ignition temperature of the thermite (1300 °C) as a result of which the exothermic reactions in the lining either did not proceed at all (mixture No. 1), or proceeded very slowly (mixtures No. 4, 5, 6 and 7), or proceeded very rapidly mixtures No. 2 and 3). When the reactions proceed at a slow rate the mass cannot be heated to a high temperature, and when the reactions proceed too rapidly the heat losses by radiation become excessive.

For the selection of a rational composition of the exothermic mixture, the temperature of the hot top lining was measured at a distance of 50 mm from the mold by means of a chromel-copel thermocouple. The junction of the thermocouple was embedded in the lining at a depth of 5-7 mm. Readings of the thermocouple were taken every 30 seconds during the time when the mold was filled up to the level of the hot top. The temperature of the hot top surface at the moment when the liquid steel was approaching it was within the range of 450-600 °C (Fig. 2).

In conformity with this result, a thermite of 400 °C ignition temperature was selected in the laboratory. The thermite contained aluminum powder (27%), sodium nitrate (15%), iron scale (56%), and cryolite (2%).

Experiments with specimens (cylinder of 64 mm diameter and 23 mm high) showed that the temperature which was attained during the combustion of the thermite constituted, depending on the composition and the amount of the refractory filler (fire brick mortar, sand, magnesite powder and fine coke were used as fillers), 800-1800 °C.

The best results (Fig. 3) were obtained when magnesite powder (40-50%) or sand (45%) was added to the thermite.

AN INDUSTRIAL ROTARY FURNACE

A.A. Garkusha and A.Z. Ryzhavskii

Giprostal'

In recent years, basically new units and technological processes for the production of steel with the use of oxygen have been developed. The following belong to this category: 1) the L-D process in which phosphorus-free open-hearth pig iron is processed in a blind-bottom converter by oxygen blowing from above and 2) two variants of the process of treating phosphorus-containing pig irons in rotary furnaces. The Cal-Do processes developed by Dr. Calling in Domnarvet (Sweden) consists of blowing oxygen from above into phosphorus-containing pig-iron in a blind-bottom, inclined, rapidly rotating convertor. The Graff-rotary process developed by Dr. Graff in Oberhausen (Federal German Republic) involves the treatment of the pig iron with oxygen in a slowly revolving cylindrical vessel called a rotor.

At present, rotary furnaces of up to 100 ton capacity are built in several countries. The first rotary furnaces, which are in operation in the Federal German Republic, are being used for processing phosphorus-containing pig iron but they can be used for pig iron of any composition.

The principle of the Graff-rotary process has been taken as the basis for the development of an experimental semi-commercial 80-ton rotary furnace for one of our iron and steel works.

The furnace consists of a cylinder with tapering ends and lined with refractory bricks (Fig. 1). Pig iron, ore and fluxes are loaded through the charging opening in one end of the furnace. Two nozzles for the supply of oxygen are introduced through the same opening. One nozzle serves for feeding oxygen into the metal for direct oxidation of pig-iron impurities, including carbon most of which is converted to CO. The second nozzle serves for feeding oxygen into the working space of the furnace for the purpose of converting the CO into CO_2 as well as for oxidizing the slag. Spouts for the discharge of steel and slag are located at the other end of the cylinder.

Fuel (gas of a high heating value or fuel oil) is used in the rotary furnace for heating the furnace before it is started after the replacement of the lining or after repairs, as well as when it is necessary to heat the charge before the addition of liquid pig iron.

The processing of phosphorus-containing pig iron involeves making two slags; the first slag, which forms after most of the phosphorus has been removed, is flushed off; the second slag is made in the furnace and it remains in the furnace after the steel pouring. Ore and fluxes for the next heat are charged on top of the slag which remains from the previous heat.

A particular feature of the process is the fact that the removal of phosphorus precedes the oxidation of carbon because at the moment of the charging of pig-iron into the furnace there is a molten active slag in the furnace and this is conducive to a high rate of the reactions and in particular of the oxidation of phosphorus. The rotation of the furnace, while ensuring the mixing of the metal with the slag, also accelerates the dephosphorization of the metal as well as assists in the cooling of the lining (which attains a high temperature during the combustion of CO) by the metal.

An early oxidation of the phosphorus in the rotary furnace makes it possible to obtain a steel of any desirable carbon content from phosphorus-containing pig iron without the need for subsequent carbon addition.

When the pig iron is blown through with oxygen, some iron is vaporized and the combustion products must be purified. It is easier to clean the combustion products in the rotary process than in the convertor process since in the former the dust particles in the flue gases are fairly large in size.

The outstanding features of the rotary process are:

1) the production of steel, not inferior to open-hearth steel, from pig irons containing phosphorus and a relatively high amount of sulfur;

2) the production of steel of any grade including alloy steels;

3) the percentage yield of sound product (up to 98%) is higher than in the Thomas process;

4) the capital costs of the erection of a rotary-furnace shop are lower than for the open-hearth process;

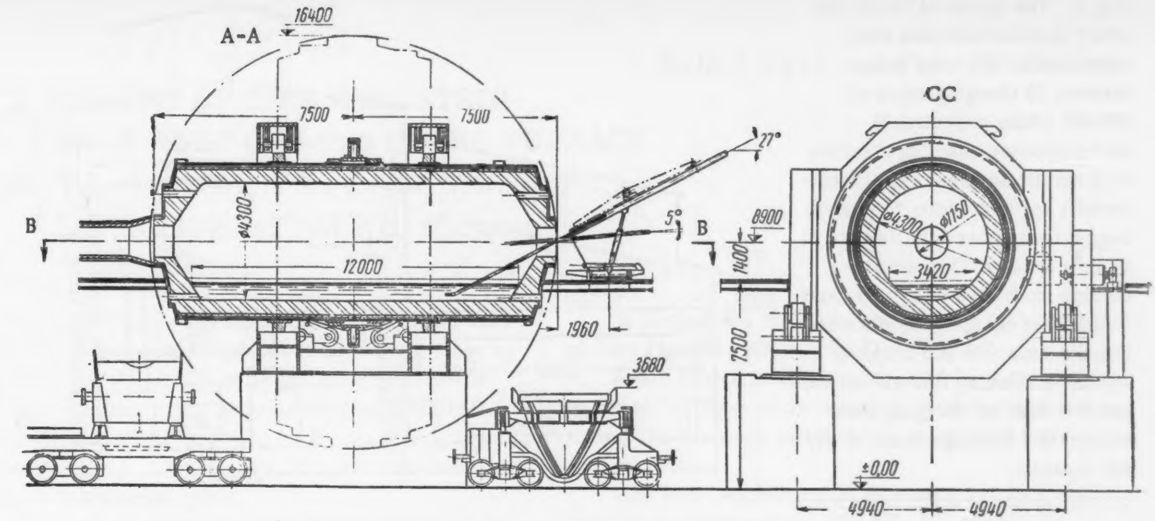
5) a better utilization of the potential heat of the pig iron which makes it possible to process considerable quantities of ore and scrap;

6) the possibility of processing pig iron of any phosphorus content, including those pig irons which cannot be used in the Thomas converter and are uneconomic to use in the open-hearth furnace;

7) a relative simplicity of the disposal of the flue gases and the absence of the contamination of the atmosphere by dust-containing gases.

A disadvantage of the rotary process is the short service life of the lining which lasts for 60-100 heats.

According to the project, the 80 ton rotary furnace is to be placed in a special building (Fig. 2). No mixer section is to be built. The pig iron is delivered to the building from the blast-furnace shop and is poured into the furnace by means of a crane of 125/30 ton lifting capacity. The same crane is used for the transfer of the steel pouring ladle from the ladle car onto the stand for steel pouring and for charging the material into the furnace.



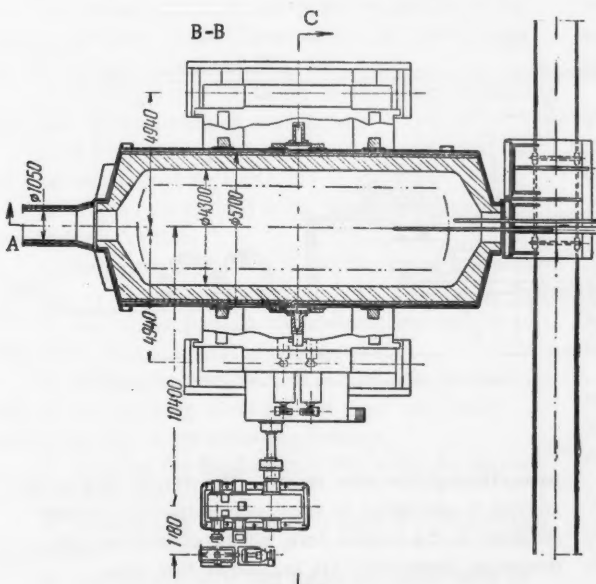


Fig. 1. 80-ton rotary furnace (project).

The ore and fluxes are charged from a transferable bin into the furnace prior to the pig iron by means of a crane for fettling the open-hearth furnace banks. Small additions of the charge materials or alloying materials are introduced into the furnace during the heat by means of a self-propelling charging machine of 2 ton lifting capacity. Stationery bins, from which materials are taken into the charging boxes, have been erected on the working floor to keep some reserve of charge materials. The bins are filled from self-unloading buckets by means of the charging crane.

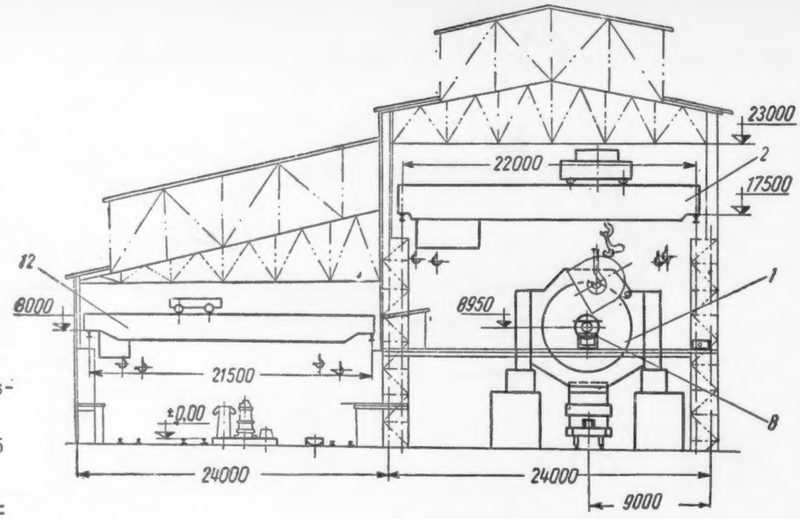
The primary phosphorus-containing slag is to be flushed off from the furnace through a special slag notch located above the opening for steel tapping at one end of the furnace or through the charging opening into the slag pot placed below the working floor on a slag car which can be driven by a locomotive or by a self-propelled steel transfer car.

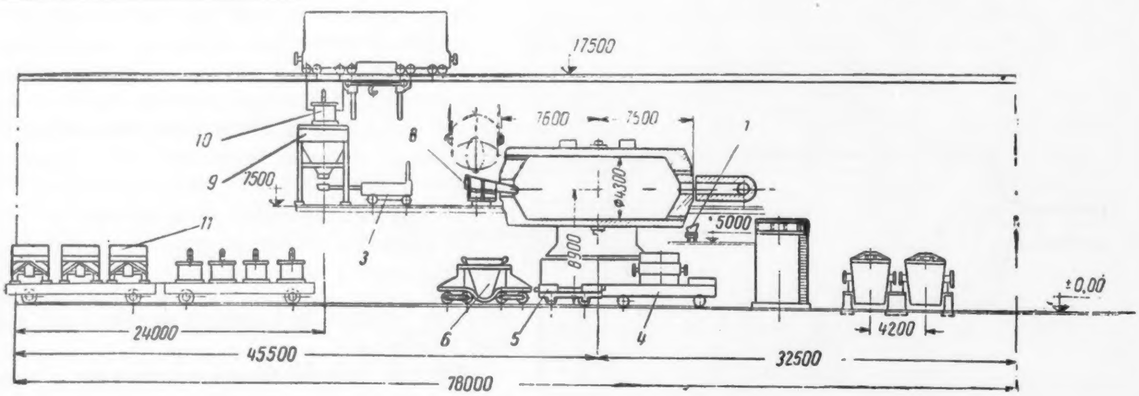
The steel from the furnace is tapped into a steel casting ladle placed on the self-propelling car which moves along the furnace on a railroad track.

Here are some design data on the rotary furnace:

Depth of the bath (liquid metal), mm	65
Quantity of slag, in % of the weight of steel	2
Thickness of slag layer, mm	200
Distance from the grade to the bottom of the charging opening, mm	725
Diameter of the charging opening, mm	1150
Diameter of the opening for the combustion products, mm	1150
Length of the furnace, m	15.0
Length of the bath including the banks, m	12.0
Furnace diameter, m	
inside	4.3
outside	5.7
Area of the metal surface, m^2	36
Thickness of the lining, mm	600
Thickness of the metal shell	40
Weight, ton:	
furnace frame	205
lining	450
liquid steel and slag (maximum)	105
furnace with steel and slag	760

Fig. 2. The layout of the 80 ton rotary furnace (side and front elevations): 1) 80 ton rotary furnace; 2) charging crane of 125/30 lifting capacity; 3) self-propelling charging machine of 2 ton lifting capacity; 4) steel-transfer car with ladle; 5) transfer buggy; 6) slag car with slag pot of 11 m^3 capacity; 7) removable tapping spout on a buggy; 8) spout for pig iron charging; 9) stationary bins for lime, ore and bauxite; 10) self-unloading buckets; 11) transportable bins for charging loose materials; 12) bridge crane of 20/5 ton capacity.





The furnace lining is made of magnesite or periclasospinelide bricks.

The furnace revolves at a speed of 0.5-2 rpm. It is driven by a 72 kw dc electric motor (with four speeds) mounted on a special platform on the right-hand side of the furnace. For the convenience of repair work on the lining, the furnace can be tilted either way through 90°. The time taken to get the furnace into the inclined position is 4 minutes. The furnace is tilted by means of a 55 kw electric motor.

A specially designed self-propelling car which moves on rails laid at right angles to the axis of the furnace is used for oxygen delivery to the furnace. Two screens which cover the front opening in the furnace are mounted on this car. Two water-cooled nozzles pass through one of the screens; one nozzle, with an 18 mm orifice, serves for the introduction of oxygen into the steel and the other, with a 38 mm orifice, into the working space of the furnace. The design input of oxygen is 1500 and 4500 m^3/hr, and the pressure in the orifice is 11 atmos and passes through the other screen. The oxygen flow to the nozzles is controlled by automatic instruments accommodated on the control desk; all the controlling and measuring instruments are located on that desk.

A lined steel flue of 1050 mm inside diameter adjoins the furnace end through which the combustion products leave the furnace. The daily loss of dust from the rotary furnace (according to data published in literature is equal to 12 kg per ton of steel, i.e., it constitutes 12 tons per 24 hr day. For the cleaning of the combustion product, a gas cleaning train which consists of a scrubber for the preliminary washing and cooling of the gas and two sprayers for the final washing of the gas had been designed.

The proposed rotary furnace, unlike the existing German furnaces, can be set up in the vertical position to facilitate repairs, and can be rotated around the transverse horizontal axis through 180° so as to achieve a more uniform wear of the lining. In this way, the campaign of the furnace can be made longer.

* * *

HOT ROLLING OF THIN WIDE STRIP ON A MILL WITH COILERS IN THE FURNACE

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TsKBMM TsNIITMASH

In August, 1957, the Soviet Union's first wide-strip mill with coilers in the furnaces, designed and made by the Novo-Kramatorka Machine-Building Factory was put into operation at the Novo-Lipetsk Metallurgical Factory. The mill is intended for the hot rolling of coiled thin strip, 1.5-5.0 x 600-100 mm, from carbon and electrotechnical steels.

The technological process of the production of the strip, as proposed by the Gipromez-NKMZ, includes the following main operations (Fig. 1):

1. Heating 2 to 3 ton ingots in four groups of soaking pits (two compartments in each group).
2. Rolling the ingots into 100-130 mm thick slabs on a universal roughing stand.
3. Cutting off the end crops, cooling, inspecting and dressing the slabs.
4. Reheating the 1.35 to 2.25 ton slabs in a continuou furnace.
5. Rolling the slabs to intermediate section, 12-15 mm thick, on a universal roughing stand.
6. Rolling the intermediate section into the final strip at the finishing, reversing four-high stand with coilers located in the reheating furnaces.
7. Winding the finished strip into a coil by means of the finishing coiler.
8. Transferring the coils to storage or to the cutting machines.
9. Cutting the coils into sheets or narrow strips by means of transverse or longitudinal cutting machines. Packing the final product.

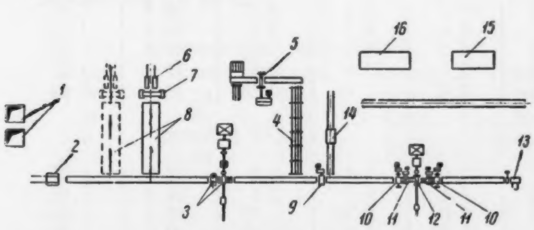


Fig. 1. The layout of the mill: 1) Soaking pits; 2) ingot car; 3) universal roughing stand; 4) roller tables; 5) 800 ton shears; 6) pusher; 7) piler; 8) reheating furnace; 9) down-cut shears; 10) coilers in reheating furnaces in front and behind the stand; 11) pulling rollers; 12) four-high finishing stand; 13) finishing coiler; 14) transfer car; 15) and 16) longitudinal and transverse cutting machines.

At present, the initial material for the mill consists of slabs 120-140 x 600—1000x 2100-2200mm.

A slab heated in a furnace to 1200 °C is transported on the roller tables to the universal roughing stand consisting of a two-high horizontalreversing stand and a vertical stand.

1200 mm long horizontal rolls are given by a reversible dc electric motor PBK 285/170 of 2950 kw capacity and speeds of 0-50-120 rpm. The speed of rolling is usually maintained between 2.2 and 3.1 m/sec.

The maximum opening between the rolls (made of 55KhN steel) is 500 mm; the rolls are mounted on textolite bearings. The upper roll is balanced by means of weights and can be adjusted at a speed of 28 mm/sec.

The vertical rolls are made of steel 50 and 900 mm long. Each roll is driven by a separate reversible dc motor MPV 42, 3/78 of 330 kw capacity and 750-1000 rpm.

The opening between the vertical rolls is 540-1100 mm; the rolls are mounted in radial-thrust conical roller bearings.

The strip is guided between the rolls of the universal stand by means of manipulators which are located in front and behind the stand and consist of two adjustable lines driven by the same motor.

The slabs in the roughing mill are rolled down to a thickness of 10-11 mm (instead of 12-15 mm as envisaged in the original design) in seven passes irrespective of the size of the initial slab.

The reduction schemes for the horizontal stand are shown in Table 1.

The vertical stand is intended for straightening the edges; the reductions in this stand usually constitute 2-4 mm.

From the roughing stand the strip is transferred on the roller tables to the down-cut shears for cutting off both ends before the strip is entered into the finishing stand.

The finishing reversing four-high stand has 1200 mm long working rolls. The back-up rolls are made of 80 KhN3V steel and are mounted on liquid-friction bearings; the working rolls are made of alloyed cast iron and are mounted on four-row conical bearings.

The working rolls are driven by a PBK-215/110 dc electric motor of 3070 kw and 100-200 rpm , the speed of rolling reaching up to 7.2 m/sec. The back-up and working rolls are counterweighed hydraulically. The speed of the adjustment of the rolls is 0.12-1.2 mm/sec.

TABLE 1

Reductions Per Pass in the Horizontal Roughing Stand

Number of pass	Initial thickness h_0, mm	Final thickness h_1, mm	Absolute reduction $\Delta h = h_0 - h_1$ мм	Percentage reduction $\Sigma \frac{\Delta h}{h_0} \cdot 100$ %	Draft $\lambda = \frac{h_0}{h_1}$
120 mm thick slab					
1	120	110	10	8.4	1.09
2	110	85	25	22.8	1.295
3	85	65	20	23.6	1.306
4	65	45	20	30.8	1.445
5	45	30	15	33.4	1.50
6	30	16	14	46.6	1.875
7	16	10	6	37.5	1.60
130 mm thick slab					
1	130	110	20	15.4	1.182
2—7	The same as for the 120 mm thick slab				
140 mm thick slab					
1	140	121	19	13.6	1.157
2	121	90	31	25.6	1.343
3	90	67	23	25.6	1.343
4	67	45	22	32.8	1.49
5—7	The same as for the 120 mm thick slab				

The reversing hot rolling of thin strip in the finishing stand is carried out with the use of special coilers set up in the reheating furnaces on both sides of the finishing stand, the coilers being intended to receive the intermediate product from the finishing stand.

Because the coilers are located in the reheating furnaces (Fig. 2), the edges and the middle of the strip should be heated up to a uniform temperature during the rolling process.

It was found, however, that the temperature of the strip fell during the rolling. When the strip was rolled in five passes the temperature drop constituted 100-150 °C, and in three passes — approximately 50 °C. No quantitative difference in the temperature of the middle and the edges of the strip was found. The temperature across the width of the strip was not quite uniform. The main parts of the coiler are a drum and a water-cooled hollow shaft. The drum, made of steel containing 25% Cr and 20% Ni, is driven by an electric motor of 100 kw capacith and 475 rpm through a single-step reducing gear of 1:6 gear ratio.

The rolled strip is entered into the furnace by means of pulling and guiding rollers. In order to prevent non-uniform heating and warping of the strip, the guiding roller is driven by an electric motor of 12 kw capacity and 660 rpm through a single-step reducing gear of 1:2.3 gear ratio. The pulling rollers also ensure that the strip is taut when it is coiled on the drum.

For entering the end of the strip into the drum slit, there is a guide, which is driven pneumatically. When the strip is entered into the coiler the guide is lifted, and when the strip by-passes the coiler the guide is lowered.

The end of the strip, fitted into the drum, bends over the edge of the slit when the drum begins to rotate and in this way the strip becomes firmly gripped for coiling. For convenience, the drum has two slits placed diagonally opposite each other.

The diameter of the drum is 1200 mm, its length is 1200 mm; the diameter of the guiding roller is 260 mm, the speed of entering the strip is 0.5 m/sec, the pulling force applied to the strip during the coiling is 200-800 kg, the temperature of the furnace is about 1080 °C, the maximum outside diameter of the coil is 1370 mm and the maximum weight is 2600 kg.

An accurate alignment of the drum in the position for entering the strip is carried out by a servomoter. For convenience, the furnace shell can be removed for inspection and repairs to the drum.

The speeds of the coilers, of the working stand and of the roller tables are synchronized so that the tautness of the strip is maintained. The strip is rolled in the finishing, reversing stand in 3-5 passes depending on the final thickness required.

In all, more than twenty strips of various dimensions can be rolled in the finishing stand. As an example, the reductions used at our Factory for rolling four strips of various dimensions are given in Table 2.

TABLE 2

Reductions in the Finishing Stand

Number of pass	Initial thickness h_0, mm	Final thickness h_1, mm	Absolute reduction $\Delta h = h_0 - h_1$ %	Percentage reduction $\varepsilon = \frac{\Delta h}{h_0} \times 100$ %	Draft $\lambda = \frac{h_0}{h_1}$
1.5×710 mm strip					
1	11.1	7.54	3.56	32.1	1.47
2	7.54	4.4	3.14	41.8	1.71
3	4.4	2.6	1.8	41.0	1.69
4	2.6	2.1	0.5	19.2	1.24
5	2.1	1.54	0.56	26.6	1.36
1,85×910 mm strip					
1	10.4	6.4	4.0	38.4	1.63
2	6.4	4.2	2.2	34.4	1.52
3	4.2	2.9	1.3	31.6	1.45
4	2.9	2.4	0.5	17.5	1.21
5	2.4	1.85	0.55	23.0	1.29
2×1000 mm strip					
1	10.0	7.4	2.6	26.0	1.35
2	7.4	5.0	2.4	32.4	1.48
3	5.0	3.25	1.75	35.0	1.54
4	3.25	2.6	0.65	20.0	1.25
5	2.6	2.0	0.6	23.0	1.30
4×1000 mm strip					
1	10.17	7.5	2.67	26.2	1.36
2	7.5	5.1	2.4	32.0	1.47
3	5.1	4.0	1.1	21.6	1.27

After it has been rolled in the finishing stand, the finished strip passes to the finishing coiler of a roller-

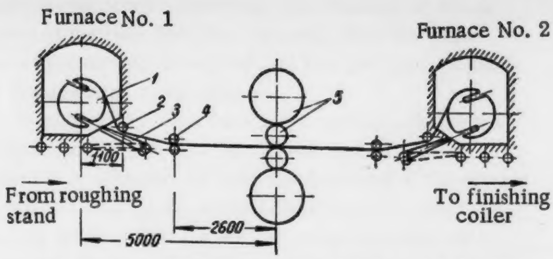


Fig. 2. Layout of the finishing stand and reheating furnaces with coilers: 1) Coiler drum; 2) guiding roller; 3) guide; 4) pulling rollers; 5) working and back-up rolls.

drum type, and then is transferred in coils to the transverse and longitudinal cutting machines in the finishing section. Depending on the designation, the coils can be cut into sheets of any desired size. The inspection, acceptance and packing of the final products are carried out in the same section.

The maximum pressure of the steel against the rolls in the roughing stand is 1815 tons for the horizontal rolls and 40 tons for the vertical rolls, and in the finishing stand it is 1260 tons. The power consumption for rolling in the roughing stand is 9-16 kw-hr/ton and in the finishing stand it is 26-40 kw-hr/ton.

The following are some operating statistics of the mill (the values are given per ton of useful product):

Electric energy, kw-hr	135
Blast-furnace gas, m^3	1060
Water, m^3	64
Metal, ton	1104
Mean output, ton/hr	40
Cost of finished sheets, rubles	617−661

The investigations carried out show that the variations in thickness of the strip rolled in the finishing stand are not always within the tolerance limits as specified by GOST.

Below, we give the results of the measurements of the thickness of sheets 2 × 1000 mm (tolerances according to GOST are ± 0.18 mm), mm:

Distance from ends of strip				
10 mm	40 mm	Middle	40 mm	10 mm
1.95	2.18	2.18	2.25	1.70
1.80	1.81	1.96	1.89	1.60
2.0	2.09	2.17	2.10	1.88
1.75	1.89	1.88	1.85	1.73

The thickness of the sheets 2 × 1000 mm over the length of the strip is as follows (thickness was measured along the middle of the sheet), mm:

Distance from the ends of the strip				
2 m	15 m	Middle	15 m	2 m
2.15	1.80	1.90	2.05	2.12
2.02	1.90	1.85	1.97	2.10
2.15	2.00	1.90	2.05	2.12
2.20	2.10	1.85	2.05	2.17
2.10	1.90	1.87	2.05	2.10

In the course of the operation of the mill certain design and technological defects have been revealed:

1) The surface of the horizontal rolls of the roughing stand scales rapidly and this results in a deterioration of the surface of the strip and leads to extra roll changes, thus reducing the mill output.

2. The coupling connecting the screws of the screw-down mechanism of the roughing mill has to be engaged and disengaged by hand. This is a cumbersome and time-consuming operation.

3. The working rolls of the finishing stand break down frequently because of cracks, the shearing of wobblers, etc.

4. The thickness and the width of the heated strip is measured by hand. This operation is inconvenient and does not produce the necessary accuracy.

5. During the rolling in the finishing stand a great deal of scale dust is produced.

6. During the transportation of the strip on the roller tables from the roughing to the finishing stand the strip is cooled substantially (its temperature drops by 60-100 °C during the setting up and cutting with the shears).

7. The front and rear cut-off ends of the strip are removed by hand from the shears.

8. The coils come out badly wound up and become flattened after the finishing coiler.

9. The marking of the coils is not satisfactory.

The shortcomings listed above should be eliminated as far as possible in the design of new similar mills. Special attention should be paid to find a method of reliably controlling the reductions in each pass in the finishing stand.

In order to reduce the temperature drop during the transfer of the strip from the roughing stand to the finishing stand it is desirable to install flying shears instead of down-cut shears for cutting off the front and rear end of the strip before it is entered into the finishing stand.

* * *

ROLLING ANGLE SECTIONS WITH THE USE OF REPEATERS

V.K. Yatsura

Deputy Director of the Central Laboratory at the Zakavkaz Metallurgical Factory

With a view to facilitating the work of mill operators and increasing the output of guide mills, repeaters, which automatically transfer the front end of the rolling piece from one stand to the next, were devised as early as 1908. These repeaters completely eliminate manual labor during rolling so that the duty of the operator consists of watching the rolling process and adjusting the repeaters and roll equipment.

The repeaters are reliable in operation if an equiaxial section square is rolled; if heteroaxial sections, such as diamond or ellipse, are transferred (especially in the finishing stand), the operation of the repeaters becomes irregular. An oval or a diamond section must be turned "on edge", as near to the vertical position as possible, before it is entered into a round or square pass; this is achieved in the trough of the repeater by means of twisting the section in the helical turning guide which is set up at the outlet from the oval or diamond passes.

The adjustment of these repeaters is difficult since apart from the design, manufacture and installation of the turning guide, some random causes which are difficult to pinpoint affect the magnitude of the final angle through which the front end of the rolled piece turns: their causes could be small changes in the dimensions of the section, changes in the temperature of the rolled piece, a displacement of the rolls, etc. In addition, the front end of an oval or a round section is slightly turned-up when it is delivered from the rolls and since the entry to the next pass is tight and sized according to the section, a slight turning-up of the front end or a small deviation of the turning angle from 90° may cause the jamming of the rolled piece.

It is not always possible to detect the true cause of the irregularities in the operation of the repeater when the mill is working, and the adjustment of the repeater frequently takes considerable time; therefore at looping mills where square sections are usually automatically transferred at the rear end of the finishing train, the oval and diamond sections are as a rule transferred by hand at the front end of the finishing train.

The difficulties in the transfer of an oval section by means of a repeater have been eliminated to a considerable extent by roller guides which have been extensively used for the past ten years.

The use of roller guides, which exert much less resistance against the entry of the front end of the rolled piece, has made it possible to adopt a repeater for oval and diamond sections.

The introduction of repeaters for the transfer of oval section has made it possible to automate fully the operation of the finishing looping trains of light-section mills at the "Dneprospetsstal", Chelyabinsk, and Kazakh Factories. Work on the adoption of this method is being carried out at the "Serp i molot", Enakievo and Zakavkaz Factories.

The production of angles. In addition to simple sections (round, square), the looping mills are used for rolling more complex sections. Until recently, all shaped sections rolled on the looping mills were handled manually at all the stands of the finishing train since it was not possible to adapt repeaters even for the transfer of a comparatively simple section such as the angle.

The adoption of repeaters for rolling angles is another achievement in the Soviet rolling technique. The first repeater of this kind was set up in 1955 between the tenth and the eleventh stands of the looping train at the light-section mill of the Magnitogorsk Metallurgical Combine. The stands were driven by two separate electric motors with a fine control of the speed so that it was possible to select a speed in those stands in such a way that the loop would not get out of the repeater and the whole roll section would pass in the trough of the repeater. But this method was not adopted by other factories because the looping mills usually have the drive on one side only and the correct discharge of the loop from the repeater under these conditions of operation had yet been not studied and was not considered feasible. For this reason even at the Magnitogorsk Metallurgical Combine the looping equipment was not used except at the 10th and 11th stands.

In 1956, repeaters were introduced for rolling angles at the light-section mills at the Dzerzhinskii Factory. The following considerations were taken into account. If the angle from the roughing stand is allowed to enter the repeater without the turning guide, then owing to a pronounced nonsymmetrical cross section, the angle when bent in the repeater, will definitely turn about and will enter the next pass either twisted or turned through a certain angle. Under these circumstances it is impossible to maintain a regular rolling process since neither the angle through which the front end twists before it enters into the next pass nor the direction of this twist can be controlled, and the section could be twisted dif-

ferently every time. However, the direction of the delivery of the loop from the repeater, after the front end of the section is entered into the next pass, depends on the direction of twisting.

Consequently, in order that the section will form a smooth loop and be ejected outside and not inside the repeater it is necessary to have the front end of the section slightly twisted in the desired direction by means of a turning guide before the section enters the repeater. However, the end of the section by continuing the twisting action in the trough of the repeater will enter the receiving pass turned through a certain angle. In view of the fact that the pass arrangement for rolling the angle is such that the passes in all stands of the finishing train are in a similar position (with their flanges downward), any twisting is not only unnecessary but undesirable.

Therefore it was decided at the Dzerzhinskii Factory to arrange the rolls of the receiving stand in the reverse position, i.e., with the flanges upward and to install a turning guide for turning the section through 40-45° near the rolls of the preceding stand. In this case, on its way from one stand to another the angle turns with its flanges upward in the trough of the repeater and enters the next pass turned through 180°.

Thus at the Dzerzhinskii Factory where the angle is rolled in three consecutive stands repeaters are installed at each mill. The rolls in the first and the last stand are set up normally, but in the middle stand the rolls are arranged in such a way that the angle passes with the flanges upward; two repeaters are used between the stands, and the rolled section is turned through 180° in each repeater.

Because of considerable difficulties due to the necessity for turning over the rolls before the installation of the repeater and also when the repeaters had to be removed, this method was not adopted at other factories.

In order that the repeater for rolling the angle can be adopted generally without the necessity for an extensive readjustment of the mill it is essential to design a repeater in such a way that the front end of the section when it is entering the repeater, will be turned up (so that the loop can be formed properly), and then the section will be turned back into the initial position before leaving the repeater.

The author of this article has developed an attachment to the repeater for turning the section so that the above condition can be met.* The attachment was used for making a turning repeater for rolling angles at the Zakavkaz Metallurgical Factory.

The turning repeater, unlike other types of repeaters, not only guides the front end of the rolled piece from one stand to another and then forms a loop, but also turns the front end of the section into the required position by means of a rigid turning attachment. A diagram of the turning repeater is shown in Fig. 1. The repeater, intended for the transfer of Nos. 2-5 angles from the sixth to the seventh (prefinishing) stand of the light-section mill, has been installed at the Zakavkaz Metallurgical Factory. So far the rolling of No. 4 equilateral angle has been mastered.

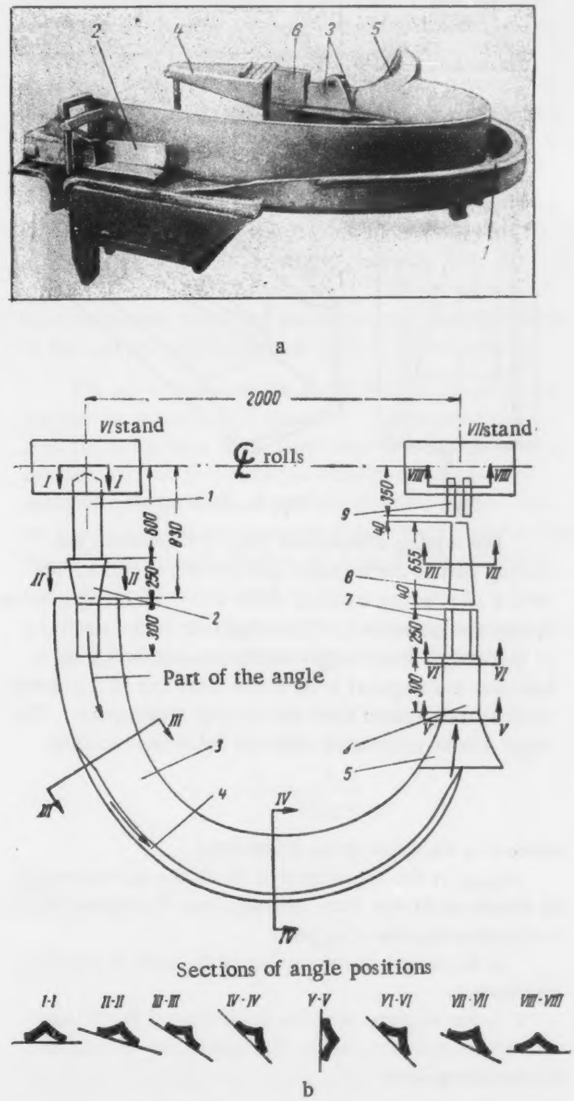


Fig. 1. Turning repeater: a) General view; b) consecutive positions of the angle; 1) turning guide; 2) guide tube; 3) trough of the repeater; 4) deflector on the outside flange; 5) loop catcher; 6) turning attachment; 7) supports of the turning attachment; 8) receiving guide; 9) entry guides.

The repeater consists of an all-welded piece of equipment with inserted cast (chromium-containing cast iron) turning attachment. The open trough of the repeater has the outside flange over its whole length perpendicular to its bottom and has a deflecting rim, 50 mm wide.

*Invention certificate No. 112206, March 13, 1957.

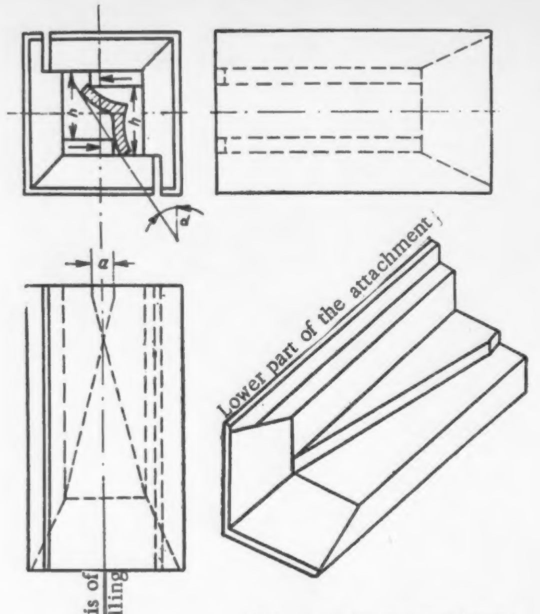


Fig. 2. Turning attachment.

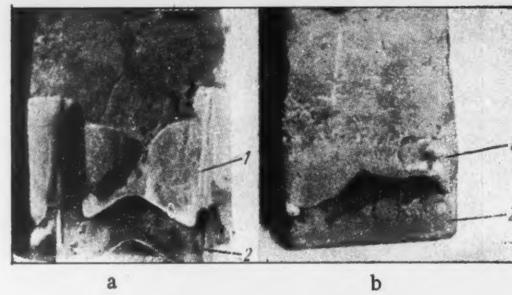


Fig. 4. Turning guide of the sixth stand: a) Entry end; b) delivery end; 1) lower part; 2) upper part.

The turning attachment (Fig. 2) constitutes the turning part of the repeater and consists of two halves with a 10 mm gap between them where adjusting wooden inserts are introduced. The magnitude of the overlap a of the wedge-shape ledges inside the attachment determines the angle of twist of the front end of the section when it is delivered from the turning attachment. The angle can be calculated from the following relation:

$$\frac{\alpha}{\alpha_{\text{total}}}=\frac{l}{L}$$

where α is the angle to be determined;

α_{total} is the angle through which the section should be turned on its way from the entry into the turning attachment to the receiving pass;

L is the length of the turning path inside the turning attachment;

L is the distance from the beginning of the turning in the turning attachment to the central line of rolls of the receiving stand.

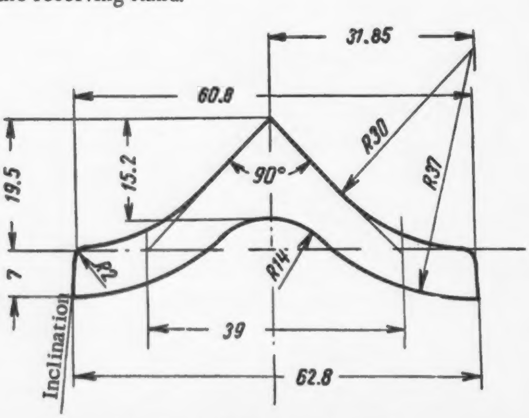


Fig. 3. Pass of the sixth stand.

The distance h is the same in both parts of the attachment and is made 5-7 mm shorter than the width of the angle. An appropriate attachment with corresponding distance h is used for each type of angle. The turning surfaces intersect (in horizontal projection) at an angle of 20-25°. Both parts of the turning attachment are fixed in a special seat of the repeater by a wedge from above and by two bolts on the sides by means of which the distance of the overlap a of the turning surfaces can be adjusted. A turning attachment of this design can be used not only for the angle but also for other nonsymmetrical sections such as strips, spring leafs, and so forth.

The receiving part of the looper has a frame for fixing a guide tube which plays an important part in the formation of the loop; without this tube it is very difficult to adjust the operation of the repeater.

The path of the angle in the repeater is quite different from the path of a square section. The angle has a higher rigidity in the horizontal plane, it begins to bend at the turning guide, and its contact with the outside flange of the repeater begins inside the trough under the rim of the outside flange so that the formation of the loop is made difficult.

An even greater difference is found in the vertical plane. Square section tends to lift the loop immediately upward whereas angle offers a higher resistance against bending upward than downward

In order to ensure a correct formation of the loop, it is necessary to make the path of the angle as similar as possible to the path of a square section which is the easiest section to handle. This is done by means of the open part of the guide tube; the vertical wall prevents a premature bending of the angle, and the horizontal wall prevents the angle from initial bending downward and forces the angle to lift the loop immediately upward The covered part of the tube makes it possible to direct the angle very accurately to the desired position in the repeater and to change this direction by means of a simple adjustment of the position of the tube without affecting the contact of the guides with the surface of the passes.

The section shown in Fig. 3 is transferred from one pass to the next by means of the repeater. It is transferred at the back of the mill from the upper level of the 6th stand to the lower level of the seventh stand located at a distance of 2000 mm (in the plane of the central lines of the rolls) from each other. The speed of the delivery of the section from the sixth stand is 4.8-5.0 m/sec. A turning guide (Fig. 4) with a turning angle of 15° is set up at the delivery side of the sixth stand; owing to this turning guide the front end of the section is turned through an angle of not more than 90° before entering the turning attachment. The turning attachment stops the turning movement started by the guide, and then turns the front end in the opposite direction so that the section enters the pass in the sixth stand again in the horizontal position.

When the turning attachment was installed, No. 4 angle was rolled at the light-section mill of the Zakavkaz Metallurgical Factory from 75 mm square billet of 120 kg weight, because of the small looping floor at the back of the finishing train, although in view of the number of working stands of the mill a billet of larger cross section could have been used.

In addition, when the rear end of the rolled piece was transferred by hand, its temperature at the time of entry into the finishing stand varied and this made the adjustment of the stand difficult.

After the repeater was installed between the sixth and seventh stands, the front end of the rolled piece could be transferred to the prefinishing stand in a very short time so that the piece could be rolled at a higher temperature. Rolling with the use of repeaters does not require a large looping space in front and behind the stand and, therefore, immediately after the first trial of the repeater (a trial period of seven days) the production of No. 4 angle from square billet of 100 mm side and 220 kg in weight was started

In the first period of the production of angle from this large billet the output of the mill at a busy period exceeded the previous maximum output by 66%. Beginning from the second quarter of 1958, the scheduled time for rolling the angle was reduced as the production of this section was considered to have been mastered.

The use of the repeater at one side of the mill and the rolling of angle with the use of looping made it possible to increase the temperature of the rolled piece substantially, to facilitate the adjustment of the mill and to improve the quality of the product. In addition, an increase in the temperature of the rolled piece during the rolling makes it possible to produce economic thin-wall angles on the looping mills.

It has been decided to install a turning repeater also at the other side of the finishing train of this mill.

* * *

TILTING MACHINE FOR INSPECTION BEDS

V.V. Trofimov

Engineer of the Department of Technical Control of the "Azovstal' "Factory

Turning railroad rails by hand on the inspection beds is a very difficult and arduous operation. In order that the surface is inspected and surface defects revealed it is necessary to turn each rail over four times. The inspection of rails is made twice: first by teams of the OTK (Department of Technical Control) and secondly by the MPS* inspector at the Factory. In addition, during the sorting and the removal of small defects the rails are again turned a few times.

With the change to the production of 25 m long rails of a weight double that of the previous type of rails, it became impossible to turn the rails by hand, in particular those of large cross section.

The turning of the rails was fully mechanized at all sections of the rail mill except the inspection beds†. It was, however, easy to mechanize the tilting operation on these beds since the rail had to be turned only once while it moved on the beds or on the rollers. For this purpose a suitable equipment for turning the rail through 90° was installed.

At the inspection beds of the "Azovstal' " Factory a turning machine designed by the foreman of the rail mill, N.N. Savchenko, was installed for the first time. The machine moves on a track laid along the inspection bed. The rails are laid on the inspection bed in such a way that the rail ends at the side of the machine can be lined up by means of a pneumatic arresting device.

The machine (Fig. 1) is mounted on a metal frame placed on two rollers. One roller (driving) is driven by

* Ministry of railroads.

† Metallurgist No. 8 (1958). [See C.B. Translation].

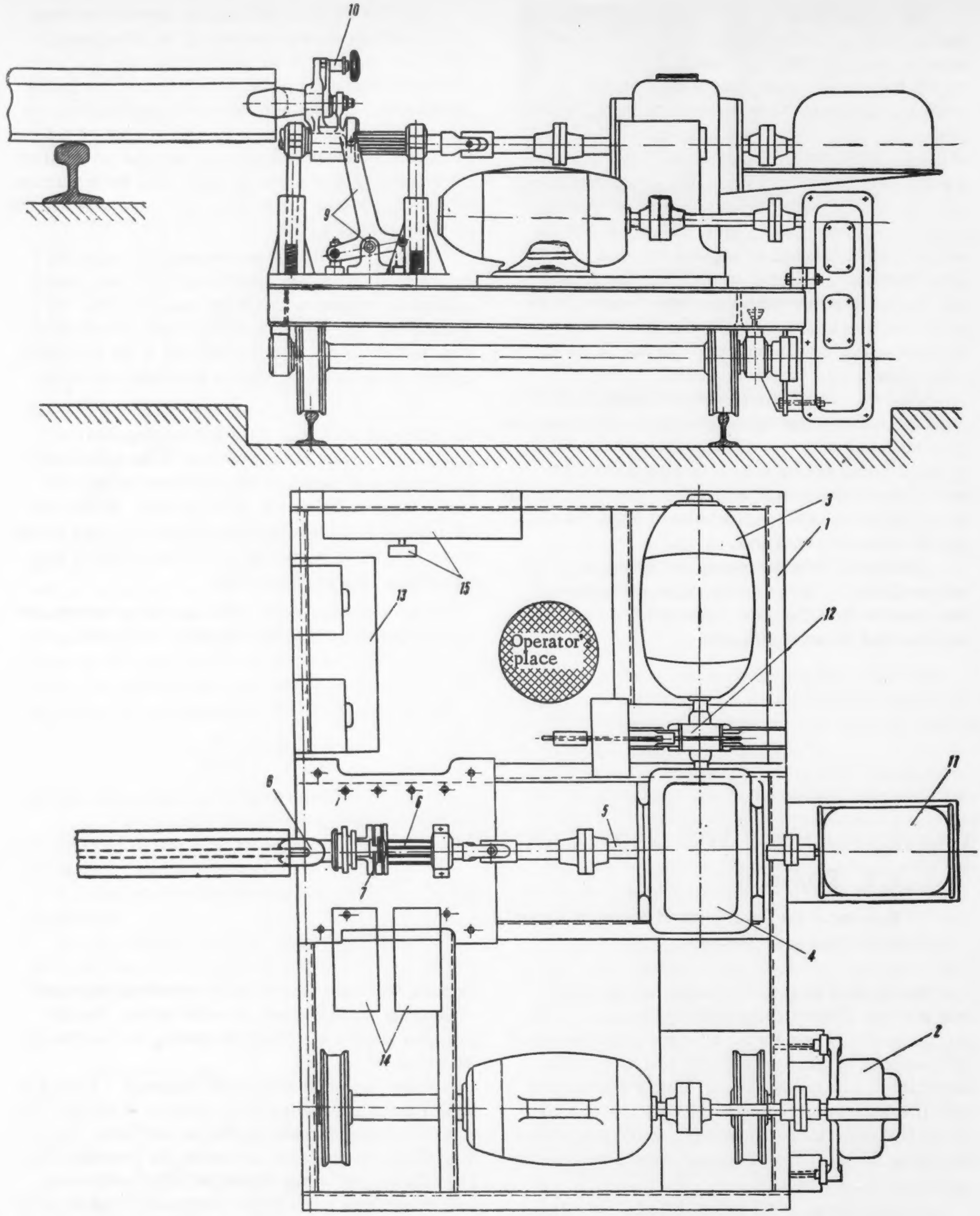


Fig. 1. Diagram of the turning machine used at inspection beds: 1) Frame; 2) transmission gear; 3) 10 kw electric motor for driving the turning mechanism; 4) worm gear; 5) driving shaft of the turning apparatus; 6) grooved shaft; 7) segment; 8) fork for engaging the rail to be turned; 9) lever system; 10) arresting apparatus; 11) end switch (servomechanism); 12) brake mechanism; 13) control panel; 14) electromagnets; 15) commutator panel.

a 5 kw electric motor via a three-speed reducing gear; the second roller is free running. The turning mechanism is actuated by a 10 kw electric motor which transmits the rotary motion to the shaft via a worm gear of 1 : 60 gear ratio. The rotary motion from the cylindrical part of the shaft is transmitted through a clutch and a universal joint to the turning mechanism. In this way it is possible to transmit the rotary motion irrespective of the inclination angle of the grooved shaft.

The turning apparatus consists of a grooved shaft and turning head (Fig. 2). The fork of the head engages the rail by means of a system of levers and then the rail is turned when the shaft rotates.

The turning mechanism is designed in such a way that the engagement of the fork with the rail can be effected in any position of the rail (with the rail resting on its head or its foot or on edge). The appropriate position of the fork is set up by means of the arresting mechanism.

The shaft is equipped with shock-absorbing bearings of helical springs for softening the impact of the rails.

One of the particular design features of the machine is the free horizontal movement of the machine in the direction in which the machine is pulled by the falling rail when the rail is turned over on one side or the other. At the moment when the rail is being turned the machine can move freely forward or backward without hindrance. No breaking devices can be used since they might cause breakdowns and damage.

The servomechanism set up on the other side of the worm shaft fixes four positions of the fork at the corresponding positions of the rail. For instance, when the rail is being turned from the horizontal into the vertical position, the servomechanism is actuated and it automatically disconnects the electric motor from the supply main and also stops the machine by means of a break mechanism at the moment when the rail comes with its base upward. The setting of the servomechanism is changed by means of an electric commutator which is mounted on the supply panel. A drum with the electric cable is located at the beginning of the inspection bed and is not connected with the machine.

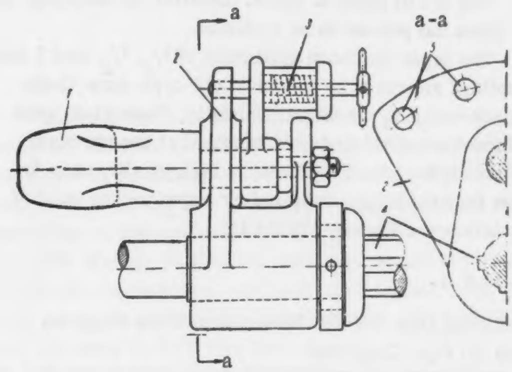


Fig. 2. Head of the turning mechanism: 1) Fork; 2) segment with the body; 3) arresting mechanism; 4) grooved shaft; 5) openings for inserting the forks.

One operator controls the movement of the machine and all the turning mechanisms.

The turning machine was found very satisfactory in operation. It is simple to operate and requires no special maintenance. The machine has a high throughput and it ensures the required speed of work on the inspection beds and adjoining sections; it ensures that the operations are carried out smoothly and at a constant rate. The machine can be used for turning rails of any dimensions.

Previously, when the rails were turned by hand, all the operations connected with the inspection acceptance were done on the same bed(accommodating 60-70 rails) and took 25-30 min.

The turning machine can carry out all these operations in 22 min.

This type of machine can be usefully employed in the rail mills at other factories.

* * *

EXPERIENCE ON MAKING COUPLINGS FROM PIPE

A.I. Fridland

Lenin Dnepropetrovsk Tube Factory

The factories which manufacture gas pipes to comply with GOST 3262-55, also make the couplings for these pipes, either by forging from a steel strip or by casting and subsequent heat treatment. Both methods are labor consuming. To make the work easier and to reduce the cost, a trial production of gas-pipe couplings from butt-welded pipes was started.

At present, all the couplings for the gas pipes of $1/2$, $3/4$, and 1 inch diameter are made from butt-welded pipes specially made for this purpose. The production of forged couplings from steel strip for pipes of $1\frac{1}{4}$, $1\frac{1}{2}$, 2, $2\frac{1}{2}$, and 3 inches diameter has also been completely discontinued in recent years. The couplings for these pipes are now made from seamless hot-rolled pipes.

Forged couplings are made now only for pipes of 4 and 5 inch diameter since no pipes suitable for these couplings are made.

The use of pipes as initial material for couplings for gas pipes has proved to be a success.

The pipes for the manufacture of $^1/_2$, $^3/_4$, and 1 inch couplings are made by means of old-type draw-chain furnace-welding machines. Initially, these pipes were welded from special-size strip rolled at the universal mill of the Factory. At present, a high-quality strip for pipes for couplings is supplied by a specialized mill of the Makeevka Factory (Table 1).

TABLE 1

Operating Data and the Dimensions of the Strips for Pipes for Pipe Couplings

Nominal diameter of the coupling, inch	Inside diameter of the coupling, inch	Dimensions of the strip for the pipes for pipe couplings, mm	Limiting of the groove of the sizing rolls, mm	Diameter of the groove of the sizing rolls, mm
$^1/_2$	18.5—18.8	86×5.0×6700	30.5—31.5	28.9
$^3/_4$	24.0—24.3	108×5.0×6700	39.0—39.5	34.3
1	30.0—30.3	137×6.0×6700	48.2—48.5	42.7
1 $^1/_4$	38.6—39.0	158×6.0×6700	55.0—55.3	50.8

After the welding and sizing, the pipes are straightened, their ends are cut off and the pipes are hydraulically tested under a pressure of 40 kg/cm^2 pressure.

Couplings of 1$^1/_4$, 1 $^1/_2$, 2, 2 $^1/_2$, and 3 inch diameter are made from seamless hot-rolled pipes produced at the factory (Table 2).

TABLE 2

Dimensions of Seamless Pipes for Couplings

Nominal diameter of the coupling	Nominal diameter of pipe, mm	Minimum and maximum outside diameter of pipe, mm	Tolerance limits of pipe wall thickness,%
1 $^1/_4$	50.5×6.0	50.1—51.1	+12.5 —10.0
1 $^1/_2$	58.8×7.0	57.9—59.0	
2	70.0×7.0	69.7—71.1	
2 $^1/_2$	88.0×8.0	87.0—88.7	
3	100.0×8.0	99.6—101.6	

The pipes are cut into pieces for couplings on a rotary cutting machine in which the carriage with the pipe is conveyed pneumatically to the rotary saw. In this way, the work of the operator is made easy and the output is increased. The length of the pieces for each coupling size is determined by an interchangeable collar inserted on the machine spindle which carries the saw. The machine has an arrester which makes it possible to fix the pipe in the required position for cutting off the desired length accurately.

The arrester is withdrawn automatically at the moment when the cutting operation is started so that the piece cut off can be easily removed. The output of such a machine constitutes 3000-3500 pieces per seven hour shift.

The subsequent treatment of the pieces cut off to produce the couplings is the same as for the coupling forgings (blanks).

The blanks obtained from the pipe have much more accurate dimensions of the inside diameter as well as a much cleaner internal surface as compared with the forgings. Therefore, it is possible to speed up the treading and to reduce the wear of the die as well as to reduce the amount of rejected material and the cost.

A fall in steel waste and a reduction in rejected material were conducive to a reduction in the cost but were not of decisive importance. The amount of waste and rejected product fell from 39.1% (of useful product) in 1952 to 31.9% in 1957. This was equivalent to savings of 715 ×(0.391-0.319) = 51.5 rubles per ton of couplings, i.e., 8.2% of the total cost reduction which constituted 628 rubles per ton. The main part of the cost reduction was obtained on account of the reduced cost of processing, which fell from 1265 rubles per ton in 1952 to 705 rubles in 1957, i.e., by 360 rubles, while the total cost reduction amounted to 628 rubles (from 2248 rubles in 1952 to 1620 rubles in 1957).

It is also significant that this substantial reduction in the manufacture cost is not a result of an increased output since the output of couplings increased by only 16% in five years (from 1957 to 1952).

Such a high reduction in cost as was achieved for the couplings was not attained for any other type of product of the tube mills at our Factory. This is a convincing testimony of the expediency of changing from forging the couplings from strip as practiced by the majority of the tube factories to making the couplings from pipe.

THE PRODUCTION OF SPECIAL DOWELS

Ya.Kh. Sartan

"Proletarskii trud" Factory

Until recently, dowels were made by a complex method, which involved machining the final product and were driven by hand into concrete or steel. At present, the dowels are mostly driven by means of an SMP-1 pistol. Therefore, the need arose for the mass production of dowels by a new method which eliminates the labor-consuming machining operation.

In accordance with the technical specifications TUM 346-57, the development of the technology and production of experimental batches of dowels was aimed at mastering the production of two sizes: 5.5 x 60 mm with a pointed end (an 8 mm diameter dowel, Fig. 1 a) and 8 x 50 mm with a pointed end and grooves cut near the head (a 12 mm diameter dowel, Fig. 1 b).

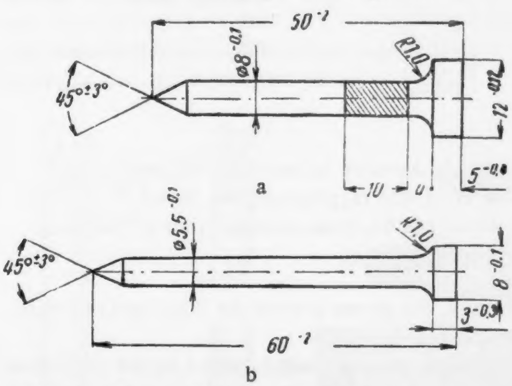


Fig. 1. Shape and dimensions of special dowels: a) 8 mm diameter; b) 12 mm diameter.

The dowels were made from 10 mm and 7.0 mm diameter wire rod (the 12 mm diameter pin and the 8 mm diameter pin, respectively); the starting material for cold upsetting was sized wire of 7.92 and 5.42 mm diameter and σ_B = 60-70 kg/mm^2.

The main operations in the production of dowels of both diameters were : pickling the wire rod; drawing the wire rod; thermal treatment (intermediate) of the wire material; pickling, washing and lime treatment; drawing to the final size; testing and inspection by the OTK; upsetting; trimming the end; finishing.

The operations in the production of 5.5 x 60 mm dowels were: grading and inspection by the OTK; thermal treatment (final); inspection by the OTK and the delivery to the storage. The operations for the 8.0 x 50 mm dowels were: groove cutting; grading and inspection by the OTK; thermal treatment (final) of the finished product; inspection by the OTK and delivery to the storage.

The wire rod was pickled at 45-65 °C for 10-20 min in 18-8% solution of sulfuric acid (the initial material for wire was pickled for 8-10 min) after the thermal treatment; washing and lime treatment was carried out according to the established method.

The drawing to the final size was carried out on a 1/700 drawing machine with a draft of approx. 10%; from 8.35 mm diameter to 7.92 mm diameter, and from 5.70 mm diameter to 5.42 mm diameter.

The wire rod of 8.35 and 5.70 mm diameter was subjected to tempering in an electric furnace at ShO-130-type at 700 °C for 1.5 hr with subsequent cooling in soaking pits.

The finished dowels were quenched and tempered. The regime of heat treatment was as follows: quenching at 840 °C, exposure for 1.5-2 min, cooling in oil, tempering at 200-220 °C and exposure for 60 min cooling in oil.

Tests on the wire and finished product gave the following values of σ_B (Rockwell test): 50-55 after the heat treatment of the wire, 60-65 after the drawing of the wire, and 46-50 after the thermal treatment of the finished product.

The upsetting of the dowels 8 x 50 and 5.5 x 60 mm was carried out on a type A-123 two-strike cold-upsetting press. Dies 70 mm in diameter and 145 mm long, an intermediate hammer of 55 mm diameter and 122.5 mm long, and a finishing hammer of 47.6 mm in diameter and 76 mm long were used. The upsetting operation and the formation of the head of the required dimensions proceeded normally. The shape and the dimensions of the dowel are shown in Fig. 2.

An LT-1 type cutting press with 16 x 22 x 30 mm cutting knives was used for cutting the end of the pin.

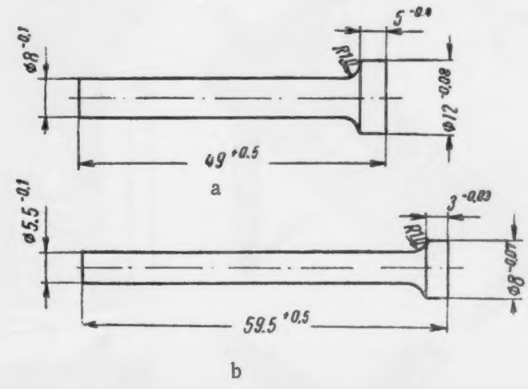


Fig. 2. Shape and dimensions of dowels after upsetting: a) 8 mm diameter; b) 12 mm diameter.

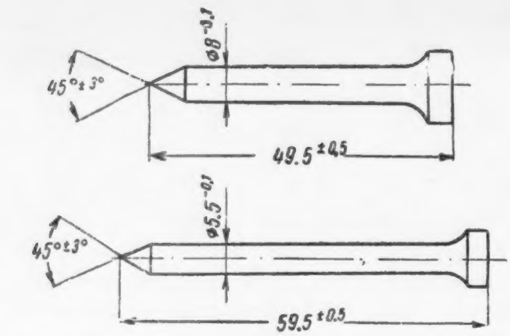


Fig. 3. Shape and dimensions of dowels after cutting.

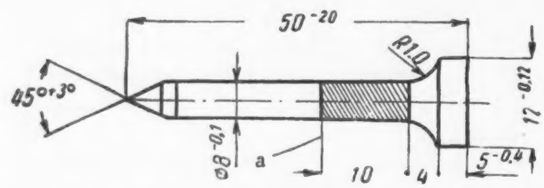


Fig. 4. Shape and dimensions of the dowel after ragging: a) Spiral grooves at 30°; pitch=1 mm; depth of the groove=0.6 mm.

The shape and the dimensions of the pointed end of the pin are shown in Fig. 3.

The ragging on the stem of the 12 mm diameter pin was cut on an A-251 automatic machine.

The cutting tools were 27 x 67 x 110 and 27 x 67 x 125 mm in size. The shape and the dimensions of the ragged portion of the stem are shown in Fig. 4. The cross section through the grooves on the cutting tools and the geometric layout of the pattern are shown in Fig. 5.

Dowels of 8 x 50 mm and 5.5 x 60 mm manufactured by this method were tested under industrial conditions by means of shooting the dowels into a target. According to the official report from a client the operation proceeded normally, the dowels pierced the target and remained firmly fixed.

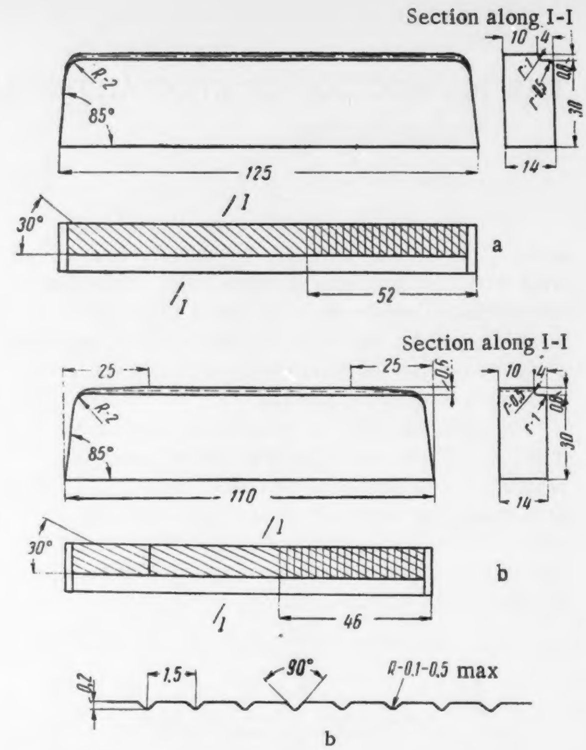


Fig. 5. Movable (a) and fixed (b) cutting-tools of the A-251 ragging machine for the 8 x 50 mm dowel and the cross-sectional view of the transverse grooves (c).

The adopted and tested method for the production of 8 x 50 mm and 5.5 x 60 mm dowels makes it possible to start the mass production of special dowels of the above dimensions which will conform to the technical specifications TUM 346-57 with regard to their mechanical properties, geometrical configuration, and the condition of the surface.

THE DEVELOPMENT OF THE WIRE-DRAWING INDUSTRY

N.K. Laman

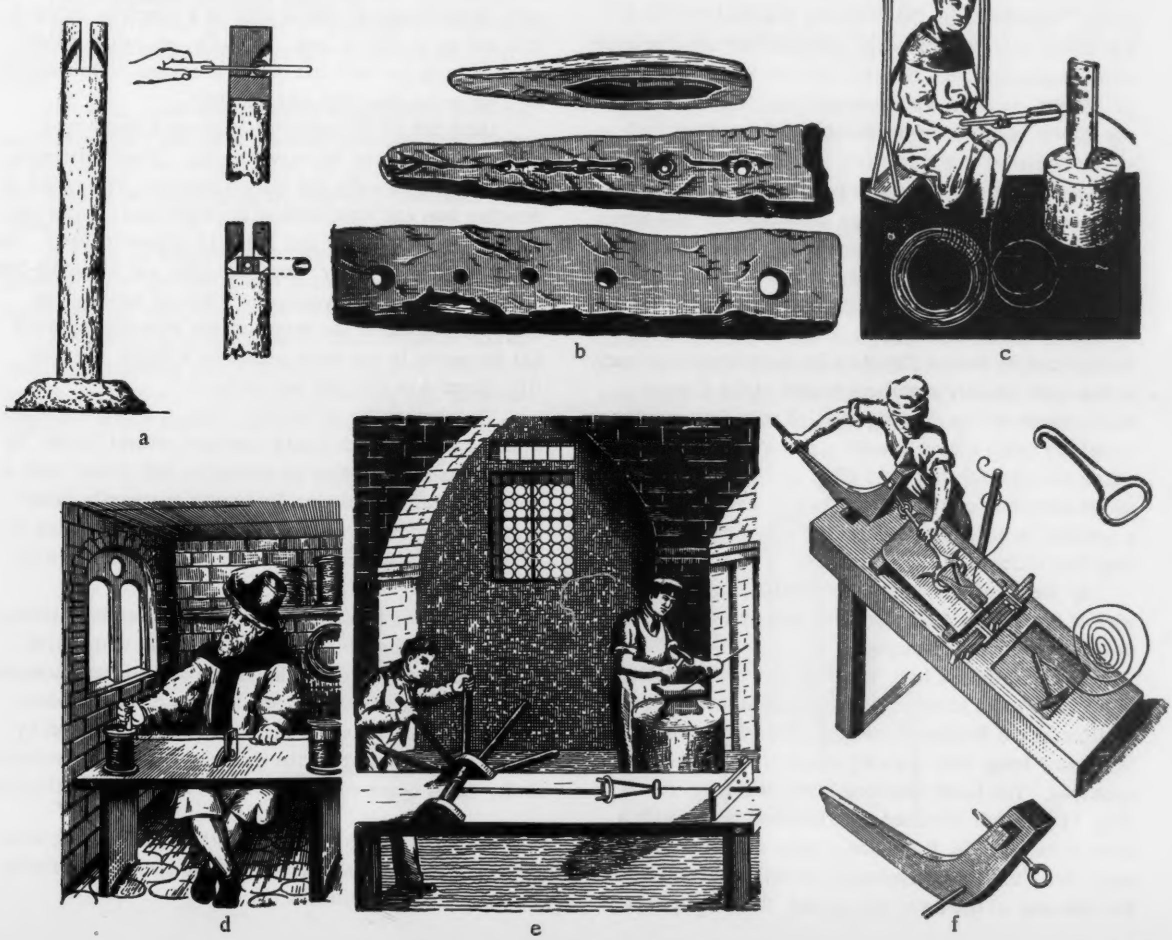
From the History of Technology

According to present day knowledge, the drawing of wire from nonferrous and noble metals originated during the third or fourth century B.C. However, even before that time the production of metal rods by hammering was known. A metal ingot was hammered and formed into a rod or flattened into a plate which was subsequently cut into narrow strips and the sharp edges were rounded with a hammer. The oldest specimens of this type were found in Assyria and they date from 4600 B.C.

The art of making wire was known in China about 2200 B.C., and in India about 2000 B.C. Objects made of drawn wire 1000 or 2000 B.C. have been found in the Soviet Union.

Reliable accounts on wire drawing from nonferrous metals in Western Europe refer to the early centuries A.D. The production of drawn iron wire began in the twelfth or thirteenth century A.D. There are some indications that the production of iron wire was started as early as the tenth century A.D.

Fig. 1. The development of the methods of making wire by hand: a) Wire drawing in Africa; b) draw plates which were driven into a log; c) swing and draw plate (fifteenth century A.D.); d) drawing a thin wire with the use of coils rotated by hand (1527); e) draw bench; f) lever-and-tongs machine (lever and ring are shown separately).



The African people developed an interesting wire drawing apparatus (Fig. 1 a) which consisted of a 600 mm high and 70 mm diameter pole dug into the earth and having a vertical slit and a deep recess for the draw plate in its upper end. The draw plate, with the end of a forged rod threaded through it, was placed into the recess in the pole, and the worker gripped the end of the wire and pulled it through the hole. Obviously, similar equipment for making wire from nonferrous metals was in use earlier among Eastern peoples. The draw plate was inserted into the recess in some kind of supporting frame or it was driven with its pointed end into a log, as indicated by the external appearance and shape of the draw plate (Fig. 1 b) found in France and dating from 400 A.D.

Various devices were employed to make the work of the wire-drawer easier. Fig. 1 c shows one of these devices which consists of a draw plate fixed in a log, and a pair of tongs attached to the worker's belt; the worker sat on a swing suspended from the ceiling or a beam. The worker first bent forward, gripped the end of the wire (which was threaded through the draw plate) with tongs and then, by leaning backward and pushing with his feet against the log, pulled the tongs and the wire. About a 0.3 m length of wire was drawn in one swing. Depending on the diameter required, the wire was drawn repeatedly through progressively smaller holes in the draw plate.

Beginning from the fourteenth centrury, thin wire was drawn by means of hand-rotated drums (Fig. 1 d) which consisted of a table with coils, which were rotated by means of handles, and a draw plate.

According to some foreign historians the first wire-drawing mill with a rotary working mechanism appeared in the fourteenth century, and a drawing mill with coilers was invented by Rudolph who lived in Germany in the first half of the fourteenth century A.D. However, investigations by Soviet historians have shown that as early as the tenth century metal craftsmen of the Kievan Russia mastered the drawing of thick wire from nonferrous metals by using a draw bench—a piece of equipment which was new for that time (Fig. 1e). The draw plate was mounted on one end of the bench, and on the other a wooden lever with a leather strip or a rope with an iron ring into which tongs were inserted.

By means of this equipment Russian craftsmen made wire of 2 to 3 mm diameter from nonferrous metals and used it for decorative purposes.

The drawing of iron, which is much less plastic, much harder and stronger than nonferrous metals, became possible in the fourteenth century when the powerful lever-and-tongs wire-drawing stands with a water drive appeared. The hand-operated lever-and-tongs stands (Fig. 1 f), which consisted of an inclined bench with a draw plate, a table, tongs and a lever connected by a steel rod to the tongs, appeared somewhat later. When the free arm of the lever was pulled, the tongs gripped the wire and pulled it through a 7-10 cm distance. These stands were used for drawing iron and nonferrous wire up to 3-4 mm in diameter from forged wire rods up to the sixteenth century.

The first water-driven stand for drawing thick forged iron rods was invented in Germany in 1351. The power of the water wheel was transmitted by large gears to the lever-and-tongs mechanism of the drawing stand.

The design of the lever-and-tongs stands remained practically unaltered for a long time.

In 1783, H. Cort invented the first rolling mill with grooved rolls which could be used for rolling long wire rod, and this made it possible to dispense with forging hammers which were used earlier for making short wire rods. Therefore, there was no more need for the lever-and-tongs stands which were replaced by much better and efficient drawing coilers.

Until the 1860's the rolling mills were used for the production of wire rod up to 6 mm in diameter and about 50 m long. The output of the mill constituted 1.5-4 tons per 10 hr shift.

The invention of the telegraph, the construction of suspension bridges, and hence the demand for steel rope, created a promising market for long iron and steel wire. In 1862, G. Bedson built the first continuous wire-rolling mill which produced rolled wire of a diameter below 6 mm and up to 200 m long. Initially, its output constituted 20 tons per shift but owing to several improvements by 1900 it reached 200 tons per shift.

Until the 1870's mainly single-draft drum-type machines were used for wire drawing. Their efficiency improved very slowly and insignificantly. The speed of drawing iron and steel wire with single-draft drums was 0.5-0.6 m/sec in 1800 and 1.0-1.16 m/sec in 1900*. The speed of drawing copper and brass wire was about 10-20% higher. Thus, in 100 years, the speed of drawing has only doubled while the output of the wire-rolling mills has increased in the same period by a factor of about fifty (from 4 to 200 tons per shift).

To increase the efficiency, wire-drawing machines should be built which would combine several blocks in one unit usually located on one table and driven from a common shaft and engine (hydraulic or steam). Wire-drawing machines with four blocks were in operation in 1812. In 1840 some works had drawing machines with ten blocks.

In 1838, in Russia G.A. Stroganov obtained a license for a drawing machine with twelve blocks with which the speed of drawing iron, steel and other wires exceeded 0.8 m/sec (Fig. 2). Cast-iron drawing blocks mounted on the horizontal shaft of the machine were driven by bevel gears transmitting the rotary motion of the vertical shaft. In its upper part each block had a slit for attaching

* A speed of 1.16 m/sec was allowable for drawing wire from cast steel or iron; the maximum speed for drawing puddling iron was 1 m/sec.

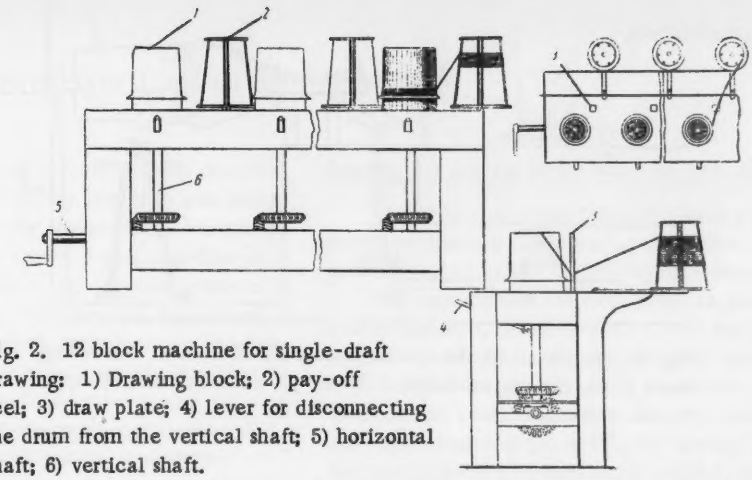


Fig. 2. 12 block machine for single-draft drawing: 1) Drawing block; 2) pay-off reel; 3) draw plate; 4) lever for disconnecting the drum from the vertical shaft; 5) horizontal shaft; 6) vertical shaft.

the end of the wire coming out of the draw plate and the pay-off reel. Any drum could be stopped by means of disengaging it from the vertical shaft by a lever.

The drawing plates were located between the block and the pay-off reel and could be lowered or raised depending on the height of the wire coil on the drum.

The Stroganov machine produced 597 meters of wire per minute and the speed of drawing was 0.83 m/sec which was 46-66% higher than the usual speed of that time.

However, even the improved single-draft drawing machines could not meet the ever increasing demands of industry. In the seventies of the 19th century the first patents for the multidraft drawing machines were taken out.

In 1871, in England, the Woods brothers took out a patent for the first multidraft stand in which the wire passed through four dies at the same time.

A machine invented by R.D. Connor (1915) was intended for drawing wire from steel, and strong and hard metals and alloys of high tensile strength and hardness (Fig. 3). It had four blocks with pins, viz., three intermediate and one finishing (receiving) block. The draw plates with the dies were accommodated near the blocks.

The wire was passed first through a soap bath, then pulled through the first die, wound around the block several times and was coiled around the block pins of the intermediate block which had a dome-shaped top. The wire was then passed over the dome and down the hollow shaft (guide tube) to the sheaves and the next die.

All the wire was collected on the finishing block. The speed of the blocks was adjusted in such a way that as the wire was removed from the finishing drum, it accumulated on the pins of the intermediate blocks.

In the 1920's tungsten carbide came into use for the production of dies. This gave a powerful impetus to the development of the wire-drawing industry, as the service life of the dies and the speed of drawing increased. Thus, for instance, nonferrous metals and alloys could now be drawn at a speed of 25-60 m/sec, and ferrous metals and iron-based alloys at a speed of 10-20 m/sec (with multidraft drawing machines). All the drawing machines, however, had one substantial drawback: 30-40% of the

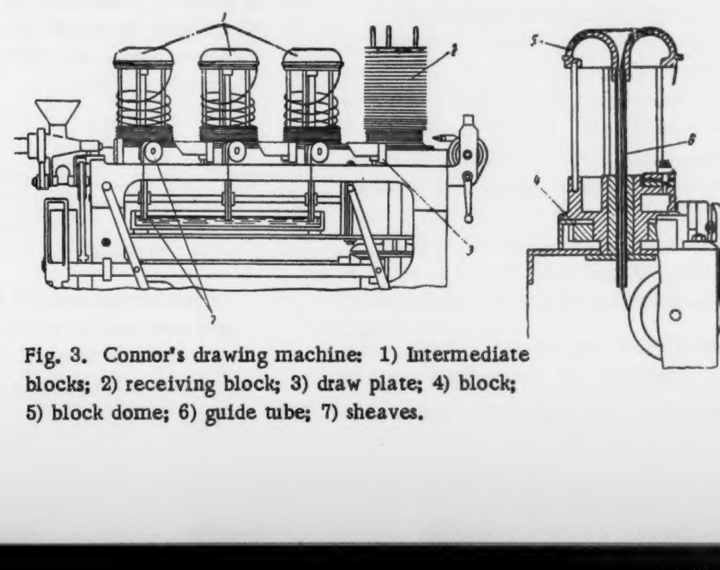


Fig. 3. Connor's drawing machine: 1) Intermediate blocks; 2) receiving block; 3) draw plate; 4) block; 5) block dome; 6) guide tube; 7) sheaves.

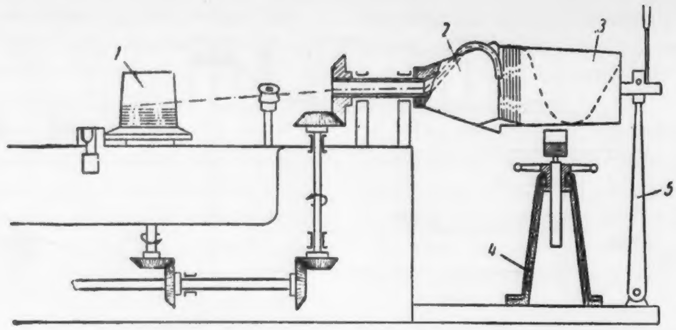


Fig. 4. Equipment for the continuous coiling of wire: 1) Receiving drum of the drawing machine; 2) "skirt"; 3) conical drum; 4) support; 5) rear-end support.

operating time was wasted on the removal of the wire wound on the drums or the replacement of the full drums by empty ones.

In 1933, a Soviet scientist, I.L. Perlin, developed a device with conical receiving drum (Fig. 4) for the continuous coiling of wire from the drawing machine. The "skirt" of this device rotates but the drum is kept stationary by means of supports, as the journal of the drum is connected with the skirt through a roller bearing. If about 4-5 meters of the wire is pulled through the die, passed over the skirt and fixed to the drum, and the skirt is rotated, then the wire begins to accumulate on the stationary drum. As soon as the drum is full, the support is lowered and the wire coil is moved further down on the drum; the support is then raised again until it touches the drum, the loose turns of the wire are bound together, the rear support is lowered and the coil is removed from the drum. During that time the skirt continues to rotate and the wire is wound on the drum.

The introduction of this attachment increased the utilization of the operation time of the machines during the production of steel wire (according to the data of the "Proletarskii trud" Factory) from 50 to 75% (25% of the time of the machine operation is used for the adjustment of the machine, the replacement of worn dies, the removal of breakages, etc.).†

A reduction in the idle time of wire drawing equipment is being achieved by increasing the service life of the dies (development of new hard alloys) and improving the quality of the wire rod.

†Continuous removal of wire with the use of so-called double drums is extensively used in the copper wire industry. The utilization of the operation time of the machine reaches 85 to 90%.

COMPACTING FINELY GROUND CONCENTRATES

The exhaustion of supplies of rich iron ores compels the iron and steel industry to utilize lean iron ores which have to be beneficiated. In the process of the beneficiation of lean ores one obtains finely ground concentrates which have to be agglomerated. The agglomeration of this concentrate by sintering is uneconomic because of the low gas permeability of the sintering charge, the low quality of the sinter, and the large losses of concentrate within the gases. Therefore, these concentrates are compacted by the method of pelletizing in granulators* and subsequent strengthening of the nodules on sintering machines or in shaft furnaces.†

Hardening hematite pellets on updraft belt machine.

At the Eagle Mills (USA), the pellets are made on plate granulators from hematite concentrate obtained by flotation and supplied by the Republic flotation plant. Since the particles are nonuniform in size and the moisture content is inadequate (6%), the concentrate, before it is fed to the granulators, is ground in ball mills and dehumidified in vacuum filters so that the final concentrate contains 9-9.5% moisture (according to American sources this is the optimum moisture content). The material is mixed with the necessary quantity of bentonite and limestone, and is transferred to the plate granulators where pellets of required size are obtained. The surface of the pellets is covered with a layer of ground coal.

The finished wet pellets are charged onto a belt machine (Fig. 1) from 4 feeders to form layers 8 inches (203.2 mm) thick. The bed is made up of roasted pellets charged from a separate bin, and of the coal necessary for ignition. The ignition temperature is 1200 °C. Air is blown from below immediately past the ignition zone.

After hardening in the heating zone, the pellets enter the cooling zone into which cold air is blown from below. The air heated by the pellets is then passed to the heating zone. The last section of the machine is intended for the final cooling of the pellets by downward induced air. The output of the plant is 2000 tons per 24 hour day‡.

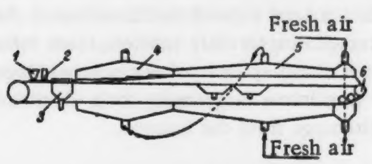


Fig. 1. Diagram of the roasting machines of the Eagle Mills (USA): 1) base-bed bin; 2) coal for ignition; 3) ignition zone; 4) roasting zone; 5) cooling zone; 6) discharge zone.

Roasting the nodules on the machine with combined draft.

The Lurgi firm (West Germany) built a combined-draft belt machine for roasting pellets from iron ore and concentrates of varied mineralogical composition.

The concentrates are compacted on plate granulators, 5 m in diameter, which produce 18-20 tons of 25-30 mm diameter pellets per hour. If necessary, water is supplied from nozzles located above the granulator. No binding additions are used. The speed of the granulator is 4 rpm and its inclination angle is 48°. By varying the speed of the granulator and its inclination angle it is possible to control the size of the green pellets.

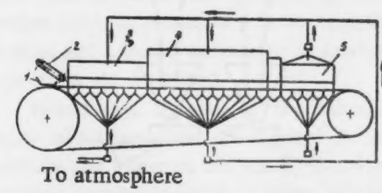


Fig. 2. Diagram of the Lurgi machine with combined blast: 1) bed feeder; 2) pellet feeder; 3) drying zone; 4) roasting zone; 5) cooling zone.

First, roasted nodules and fragments, less than 15 mm in size, are charged onto the belts to form the bed. The machine is divided into three zones: drying, roasting, and cooling. The direction of the hot and cold air flow is shown in Fig. 2.

The pellets are roasted by means of fuel-oil or gas burners. In this way, it is possible to produce any desirable atmosphere (oxidizing, reducing, or neutral) in any required point of the machine.

Roasting the pellets in the shaft furnace.

The method of roasting in the shaft furnace is characterized by a small fuel consumption and low capital outlay. At the Erie plant, pellets with coal are produced in drums which have one end made in the form of a screen with 15 mm openings. Raw pellets which pass through the screen are returned into the drum. 0.4-0.7% bentonite is added to the charge to increase the strength of the dry and roasted pellets. The pellets from the drum are transferred to the shaft furnace of rectangular cross section (1.83 × 4.27 m) and about 14 m high (Fig. 3) which operates continuously. At any time there is approx. 200 tons of

*N.A. Yarkho, Metallurgist, No. 8 (1958), [See C.B. Translation].

†Mining Engineering, No. 3 (1958).

‡ 3. In this article the "short ton" equal to 907 kg is used.

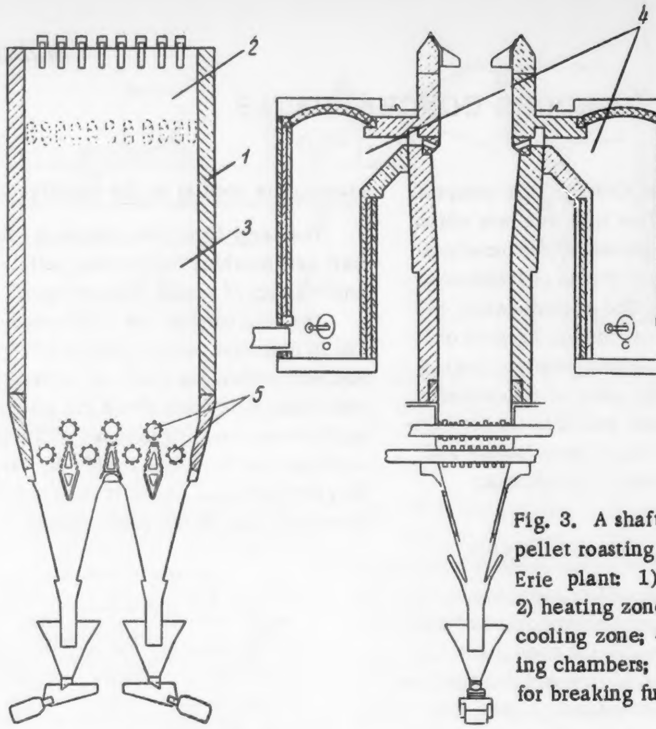


Fig. 3. A shaft for pellet roasting at the Erie plant: 1) furnace; 2) heating zone; 3) cooling zone; 4) heating chambers; 5) rolls for breaking fused chunks.

pellets which descend into the furnace at a speed of 12-25 mm/min. The furnace operates on the counter-current principle: as they descend, the pellets are heated not only by the gas which comes from the heating chamber but also by the air which is heated by the roasted pellets. Toothed rolls mounted at the outlet from the furnace break down fused chunks.

The heating chamber, which accommodates the fuel-oil burners, receives about 35% of all the air which is used in the process. The temperature of the gas entering the charge is about 1300 °C.

After being discharged from the furnace, the pellets are screened on a sieve with openings of about 5 mm. The fines are returned into the cycle.

The most economic of all the methods discussed above is roasting in shaft furnaces. It has the following advantages.

1. The output of the shaft furnace described reaches 1200 tons per 24 hour day. If calculated on the basis of 1 m^2 of the area of the unit, this is the highest output of all the methods discussed above

2. The content of FeO in the roasted pellets discharged from the furnace constitutes 0.1-2.4%. These pellets can easily be reduced whereas pellets produced on belt machines contain 18-20% or more FeO and their reduction in the blast furnace is rather difficult.

3. Repairs to the shaft furnace do not involve much labor and do not require large quantities of steel as compared with the repairs of belt machines.

4. Capital and operating costs per ton of pellets roasted in a shaft furnace are considerably lower than per ton of pellets roasted on a belt machine.

5. The working conditions of operating a shaft furnace are much better than for belt machines.

In the design of a shaft furnace and in the development of technology of roasting one must, however, keep in mind the fact that in the USA no limestone for fluxing the pellets is charged into the shaft furnace. Therefore, during the roasting process, high-melting-point silica slags are formed and the sintering of separate pellets into large chunks does not constitute a problem if a breaker is provided in the lower part of the furnace.

If this method is used without modifications in the USSR, where fluxing is extensively applied, then, owing to the formation of a large quantity of easily fused slags, the pellets may fuse into a large bulky mass which would be difficult to discharge from the furnace.

R.S. Livshits. The Distribution of the Iron and Steel Industry of the USSR. Academy of Sciences, USSR Press, Moscow, 1958, 375 pages.

Since N.F. Berezov's book, published in 1933 and by now somewhat out of date, Soviet economic literature had no survey publications devoted to the problem of the distribution of iron and steel industry of the USSR. Therefore the appearance of R.S. Livshits' book, which to some extent fills this gap, must be welcomed.

The author analyzes the specific features in the distribution of the iron and steel industry from the scientific and economic point of view, he shows the advantages of the socialist method of distribution of the iron and steel industry as compared with the capitalist method, and he describes ways of improving the existing distribution of iron and steel factories in the USSR; the history and many problems of the development and distribution of ferrous metal production in the future are discussed. Some problems related to methods of determining economic expediency when selecting districts and localities for the establishment of the iron and steel industry in the USSR are also discussed.

The book is written clearly and is well laid-out and edited. However, it has some shortcomings. Thus, for instance, while correctly interpreting the false premises of foreign economists in relation to the theory of the distribution of industry, R.S. Livshits unfortunately confines himself only to the critical analysis of old and well-known conceptions without giving any attention to modern views and works of foreign theoreticians.

Not enough use is made of the index system which would show the effectiveness of the changes in the distribution of the iron and steel industries in the USSR. Essentially, there is no analysis of the capital outlay in the iron and steel industry for separate large economic geographical regions of the country, and there is no comparison of such factors as the recovery of capital outlay and the economics of industrial establishments in various regions.

The problems of the future development of the distribution of work between the USSR and other Communist block countries in the field of the iron and steel industries are not touched upon, even in general outlines. While describing the achievements in the development and the distribution of the iron and steel industry in the USSR, the author simply omitted to mention the people who actively contributed to these achievements. The book contains no illustrative maps.

These shortcomings do not make R.S. Livshits' book less valuable, but the author should take these aspects into account when preparing a new edition of the book.

N.M. Oznobin and A.B. Rozentreter

B.P. Bel'gol'skii and A.L. Starosel'skii. Increasing the Efficiency of Rolling Mills. Khar'kov, Metallurgizdat, 1957, 183 pages.

The book contains a description of some practical methods of increasing the output of rolling mills. The methods are connected mainly with revealing internal production reserves and are based on experience; methods of increasing the output of blooming, rail-structural and section mills by means of a better organization of production and a better utilization of existing productive capacities are described.

In addition, the book gives a calculation of the mean output of rolling mills, the rational distribution of reductions and passes in the stands, the selection of the optimum weight and size of ingots and billets and production planning in rolling mills.

The book is intended for steel-rolling engineers and can also be useful for technical personnel at rolling mills and for university students.

S.S.

D.G. Ignatenko, A.L. Starosel'skii and V.B. Perchanik. The Work at the Control Post of the Rolling Mill. Moscow, Metallurgizdat, 1957, 246 pages.

The book contains general information on the properties of steel; the theoretical principles of the rolling process are discussed; the design of rolling mills and auxiliary equipment in rolling mills is described; the elements of roll design, the organization and the technology of rolling, and the organization of work and the working place are discussed.

In addition, there are short accounts of electrical engineering and electric equipment for drives and auxiliary mechanisms in rolling mills.

The publication constitutes a text book for improving the knowledge of operators at the control posts of rolling mills and is also intended for students in technical schools.

S.S.

M.M. Dizhur, N.G. Starovinskaya and E.I. Sushkina. Blast Furnace Production. Bibliography. Edited by Eng. N.B. Arutyunov, Moscow Metallurgizdat, 1958, 319 pages, (free on demand).

This guide to the bibliography of blast-furnace production contains a full literature survey and a well-worked-out classification system of the material it deals with. No doubt this is a great success for the compilers of the survey for which a need has been felt for a long time by specialists in blast furnace production and librarians.

However, in spite of an almost complete survey of the literature concerned, there are some unfortunate omissions. For instance, there is no mention of the majority of L. M. Sapozhniko's works on the quality and mechanical strength of coke, on mention of the translation of the book "Coke for Blast Furnaces" by Moo and Willer, the book "The Irregularities of Blast Furnaces" by Willcox (published by Gipromez, 1933), and others. The aim of the editors was, first of all, to give a survey of Soviet literature and therefore the reviews of foreign literature published in the Journals "Domez" "Metallurg", "Stal'", "Sovetskaya metallurgiya", "Novosti inostrannoi metallurgii", "Problemy sovremennoi metallurgii" and other journals, are not mentioned. It would be useful if this literature guide contained references to articles on blast-furnace production which appeared in the Large, Small, and Technical Encyclopedias.*

The above shortcomings do not, however, reduce the value of this literature guide.

M. Ya. Ostroukhov

*Soviet Encyclopedias.

SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosénergoizdat	State Power Engr. Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LÉIIZhT	Leningrad Power Inst. of Railroad Engineering
LÉT	Leningrad Elec. Engr. School
LÉTI	Leningrad Electrotechnical Inst.
LÉTIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MÉP	Ministry of Electrotechnical Industry
MÉS	Ministry of Electrical Power Plants
MÉSÉP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTi	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NIIZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOÉ	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIÉL	Central Scientific Research Elec. Engr. Lab.
TsNIÉL-MÉS	Central Scientific Research Elec. Engr. Lab.-Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIÉSKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZÉI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us – Publisher.